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INFORMATION CIRCULAR

MINING METHODS AND COSTS AT EL POTOSI MINE, EL POTOSI MINING CO., CHIHUAHUA, MEXICO

BY

HARLAN A. WALKER

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By Harlan A. Walker[2/]

INTRODUCTION AND ACKNOWLEDGMENTS

This paper on the mines of El Potosi Mining Co. is one of a series being prepared by the United States Bureau of Mines on mining practices, methods, and costs in the various mining districts of North America.

Grateful acknowledgment is made of the helpful suggestions and criticisms offered by the management, together with the assistance of the mining staff in preparing maps and drawings and of C. L. Thornburg, mine geologist, in preparing the section on geology.

LOCATION

The mines of El Potosi Mining Co. are at Santo Domingo in the center of the Santa Eulalia district, State of Chihuahua, Mexico, and are about 16 miles by automobile road almost due east of Chihuahua, the State capital (fig. 1). The main camp at Santo Domingo is 6,000 feet above sea level and is situated in the principal canyon about a mile above Santa Eulalia (recently renamed Aquiles Serdan), the mining town from which the district derives its name. This town, which has a population of 9,000, was founded early in the 18th century following the discovery of the ore deposit in 1703. It is the seat of the local municipal government and is typical of Mexican mining communities. The mining district comprises an area about 9 miles wide and 7.5 miles long and includes the principal part of the Santa Eulalia Range, which rises abruptly 2,000 feet above the plains to an elevation of more than 7,000

1/ The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6804."

2/ One of the consulting engineers, U. S. Bureau of Mines, and mine superintendent, El Potosi Mining Co., Chihuahua, Chih., Mexico.

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feet above the sea level. The topography is rugged and the climate semi-arid; annual rainfall averages 15 inches and occurs in summer.

The district is readily accessible by train, airplane, or automobile from El Paso, Tex., 225 miles north of Chihuahua; from the latter it is about a 45-minute automobile ride to the mines. Chihuahua may also be reached by train or airplane from Mexico City, 1,000 miles southward. Ore, concentrates, and supplies are transported over the company's 16-mile 30-inch-gage electric railroad; the upper terminal is at Santo Domingo and the lower on the National Railways of Mexico at Hacienda Robinson, 3 miles from Chihuahua, where the head offices of the company in Mexico are located. A branch line connects with the American Smelting & Refining Co. smelter at Avalos, 6 miles from Chihuahua. A 36-inch-gage steam railroad, the Mineral runs intermittently between Chihuahua, the smelter, and Santa Eulalia, where the lower tramway terminal of the American Smelting & Refining Co. (mining department) is located.

The Cia. Minera de Penoles, S. A. (Mexican subsidiary of American Metal Co.), which also has extensive holdings in the district, is served by a broad-gage branch line from the National Railways of Mexico to a terminal below Santa Eulalia, from which point a tramway extends several miles to the various mines.

HISTORY

It is generally accepted that ore of commercial value was first discovered in 1703 by three outlaws who had been expelled from the Chihuahua diocese, although according to legend Spanish prospectors made bonanza discoveries as early as 1591. Records show that more than $100,000,000 worth of silver was produced from 1703 to 1800; from the beginning of the 19th century to the present time several hundred millions additional have been produced. The early mining methods were crude and were concerned mainly with the bonanza, oxidized silver ores. After 1880, with the advent of foreign capital, principally American, improved methods of transportation, extraction, and supervision increased the scope of mining and immediately made accessible many deposits hitherto too remote or low grade for successful exploitation. As a result, the first 30 years of the 20th century undoubtedly surpassed any corresponding period in the camp's history in tonnage and net value of ore produced.

The principal mining companies in the district are El Potosi Min Co., American Smelting & Refining Co., and Cia. Minera de Penoles, S.A. The El Potosi Mining Co. controls a total of 165.13 pertenencias (408 acres); the main block of ground comprises 92.75 pertenencias (229 acres) in the heart of the district. Interests allied closely to the present owners of the mines first entered the district in 1881; thereafter operations were carried on successively under several corporate names, the last of which i El Potosi Mining Co., subsidiary to the Howe Sound Co.

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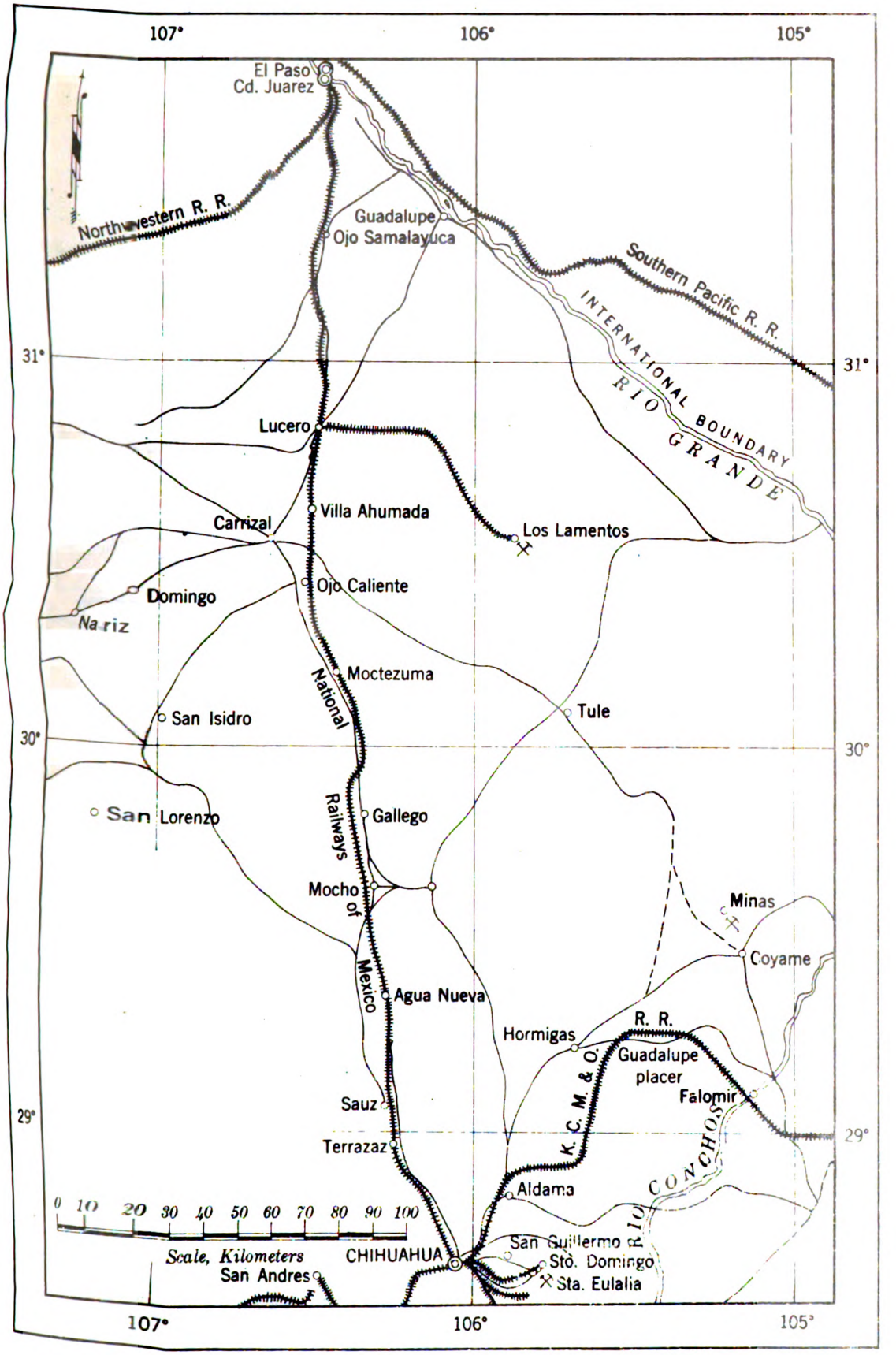

Figure 1.— Map of Northern Chihuahua, with special reference to Santa Eulalia mining district.

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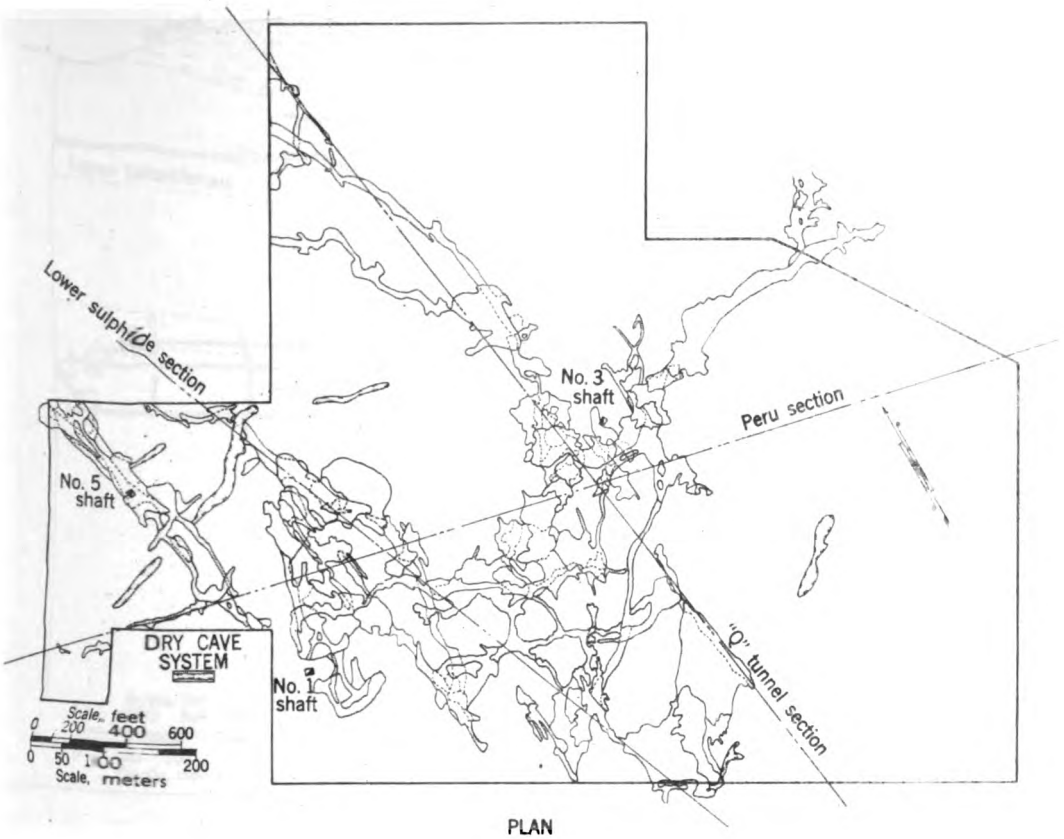
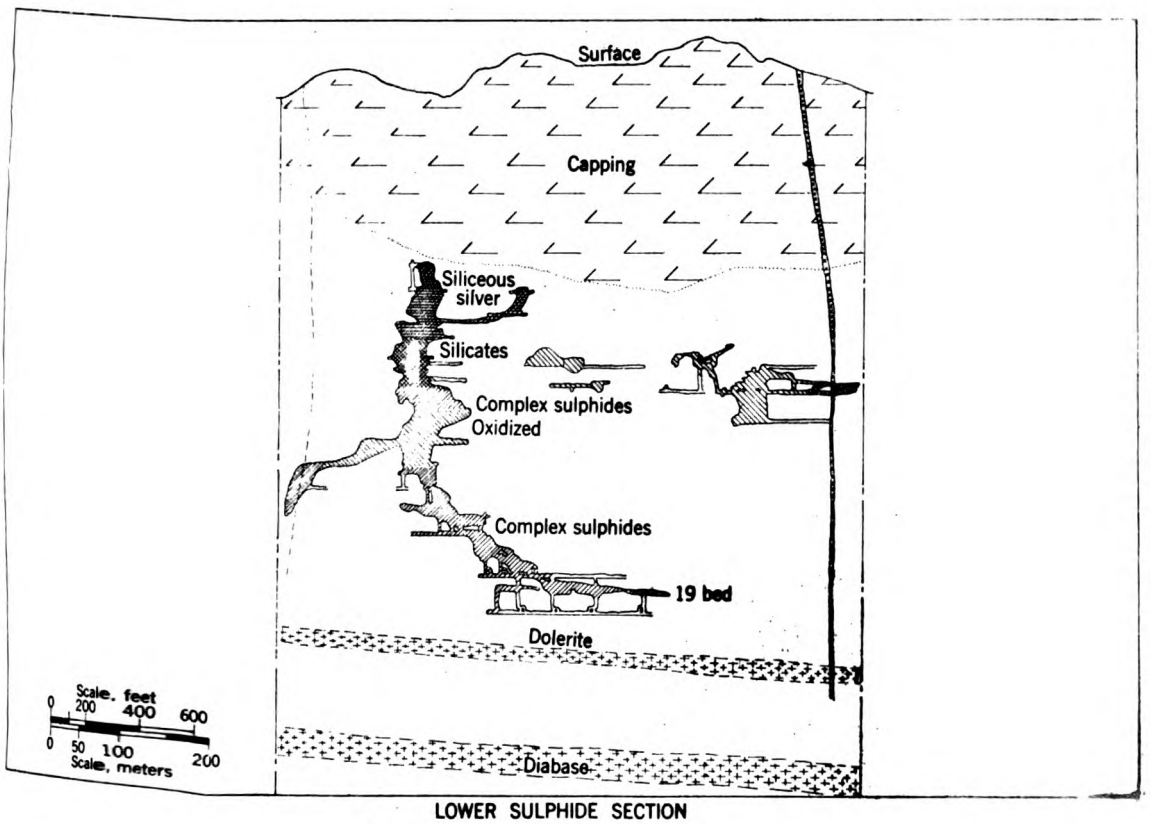


PLAN



LOWER SULPHIDE SECTION

Figure 2A.—Plan and sections of ore bodies.

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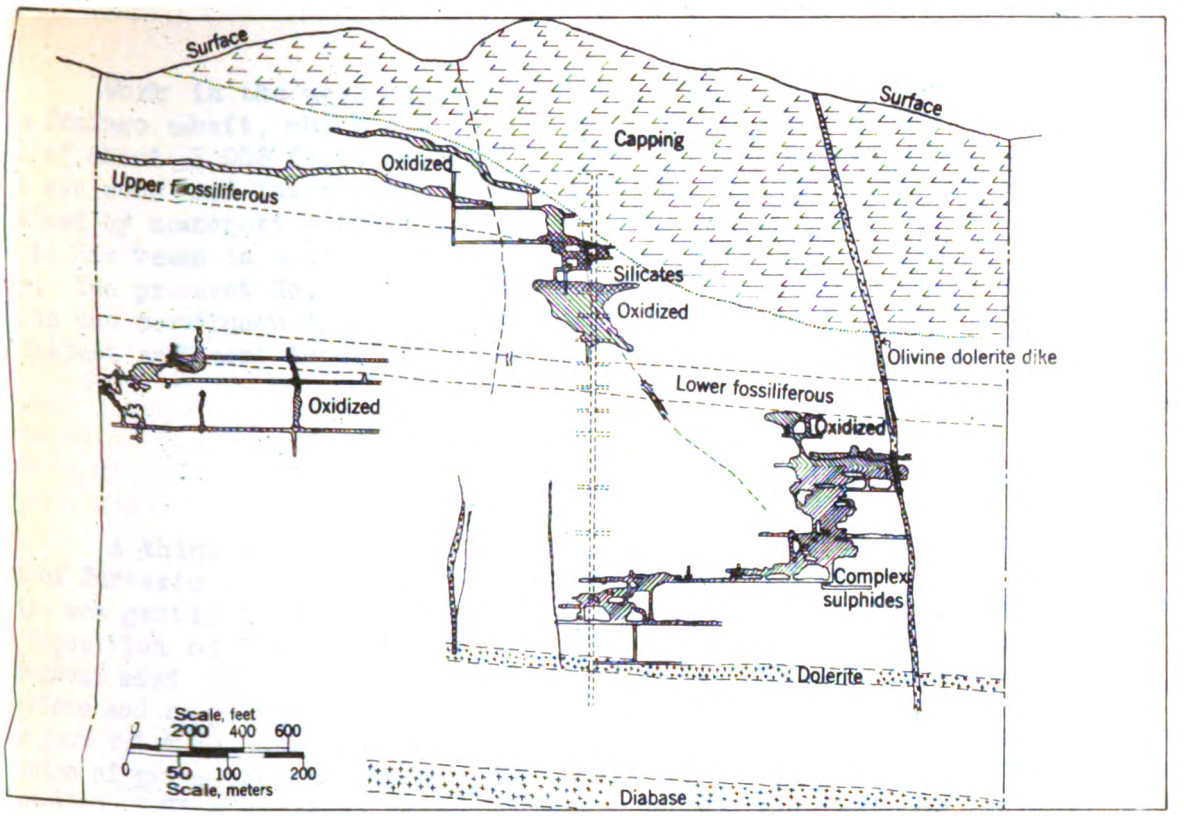
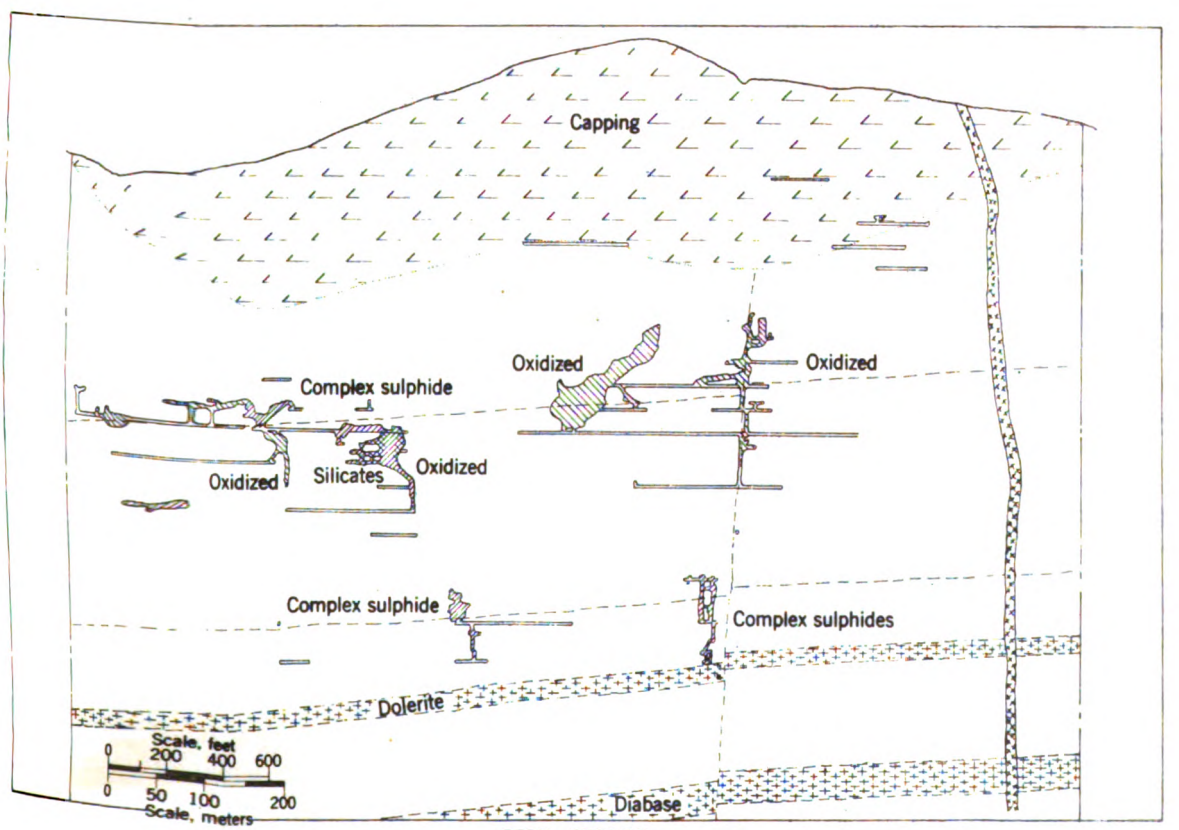


"Q" TUNNEL SECTION



PERU SECTION

Figure 2B.—Sections of ore bodies (continued).

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Work in the main mineral zone began with the sinking of the old Santo Domingo shaft, which cut into a large chimney of carbonate ore at a depth of about 1,000 feet. Following this the present Santo Domingo or No.5 shaft was sunk and later abandoned because of a bad cave-in. In 1929 it was reclaimed by concreting through about 300 feet of bad ground; since that time it has been in active service to considerably greater depths than formerly. The present No. 1 shaft was sunk on El Potosi ground in 1901 and has been in use continuously since that time, except for brief shutdowns. It is the deepest and most important shaft in the district.

GEOLOGY

General

A thick series of limestones, mostly of Cretaceous but probably in part of Jurassic age, constitutes the dominant rock of the district. This series was gently folded, uplifted, and eroded to a rugged topography before the deposition of Tertiary volcanics, which form a covering or "capping" throughout most of the district. Small postore doleritic dikes cut both limestone and capping and pre-ore rhyolite dikes cut the limestone and the lower part of the capping. Pre-ore doleritic and diabasic sills and a large intrusion of pre-ore rhyolite, probably also a sill, have been found by deep exploration. The greatest known thickness of the limestone is 3,150 feet. Structural features are not pronounced; and there is no faulting of consequence, except in the eastern part of the district. In the southwestern or main section, the part under discussion, the limestone beds generally strike about N. 60° W. and dip 5 to 10° southwest, forming the southwestern flank of a slightly domed structure, the major axis of which strikes north.

The ores are of the replacement type and occur as irregular bodies in the limestone in both pipe and tabular form. The two principal classes of deposits are locally known as "chimneys" and "mantos", the chimneys being those that cut through the strata and the mantos those that follow certain beds (fig. 2, A and B). Commercial deposits occur only in the limestone, but on and near the surface silver ores frequently are found in contact with and below the capping.

Fissures

Premineral fracturing of the limestone caused innumerable small fractures, which are most abundant and strongest in certain zones and have a predominant northward trend. Another less conspicuous system strikes northeast, and a third class, as a rule still less pronounced, strikes northwest. Deposition of ore was influenced by these premineral fracture systems. Postmineral fractures are often found parallel to and near the premineral fracture zones, especially where deposition of ore has been strong.

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Ore Channels

Ore channels occur in all premineral fracture systems but are associated more frequently with those of a northward trend. The large chimneys always are identified with that class, although their location may be influenced by other structural features as well. Mantos occur principally in certain favorable limestone beds and premineral fracture zones. The more crystalline members of the upper and lower Fossiliferous limestones, which were at one time regarded as almost the only favorable strata for mantos, appear to have been crushed and brecciated to a greater extent during folding and the following adjustments than the less-crystalline beds and were rendered more amenable to replacement thereby. The amenability of certain strata to replacement generally is attributed more easily to physical than to chemical causes. Other types of beds in El Potosi have been found to be very important in recent years. For example, shaly seams and thin, impervious dolomites seem to have had a damming influence, which resulted in extensive replacement of the limestone below them.

Mantos sometimes meander for great distances at low angles from the horizontal and either may be branches from a chimney or the result of the chimney itself being deflected to conform with and follow a certain bed. Where a manto steps across beds for any considerable vertical range it in turn assumes a chimney form. Less-extensive mantos reaching out only short distances from chimneys are common.

Ore Deposits

The attitude and form of the ore deposits are influenced by various combinations of the above factors; consequently, mining methods frequently are subject to corresponding changes. Both chimneys and mantos are highly irregular in size and shape. The great cross-sectional area and vertical range of the more important ones is outstanding in the realm of limestone replacement deposits. A typical large chimney, with a cross-sectional area of 10,000 square feet, may extend vertically 1,000 feet or more. Several of the larger mantos are more than 100 feet wide and 10 to 20 feet thick, but those of great length (in some instances thousands of feet) are only 30 or 40 feet wide and generally thinner. One sulphide manto now being worked has a known length of 750 feet and a width of 75 to 150 feet, and is 60 feet thick. This manto is on the favorable no. 19 bed and is overlain by an impervious dolomite. The extraordinary thickness probably is due to the damming influence of several intercalated, thin, shaly beds which closely underlie the dolomite. Prolonged exploitation of the mines has revealed that the ore deposits are remarkably continuous, notwithstanding their tortuous and erratic nature.

Mineralization

Two principal types of mineralization that occur are roughly confined to different vertical ranges. These are represented by commercial deposits that generally exhibit continuity throughout their respective vertical ranges. In some cases their overlapping displays ore in commercial quantities

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from the surface to the deepest known occurrence of ore.

Mineralization is especially strong in El Potosi mine and is of several distinct types whose interrelationship is too extensive to be discussed in this paper. Virtually all the production now comes from the lower part of the mine, where the ore is a silver-lead-zinc sulphide, an intimate mixture of crystallized, medium-grained pyrrhotite, marmatite, galena, and pyrite. The pyrrhotite represents 35 to 40 per cent of the total weight and is magnetic. The zinc values are in marmatite, which when pure has the approximate formula FeS.3(ZnS). The main portion of the silver is associated with the galena. These ores have a vertical range from generally within 1,000 feet of the surface to below the present lowest workings. Oxidation resulted in a silver-lead carbonate ore in the upper part of this zone. The following are typical analyses of unoxidized and oxidized ore of this kind:

		Percent				
	Ag, oz.	Pb	Zn	Insol.	Fe	S
Unoxidized	8.1	10.5	11.2	2.8	28.0	26.4
Oxidized	15.0	13.0	2.5	5.0	28.0	2.0

	Percent			
	Mg	Mn	Cu	Cd
Unoxidized	0.13	0.6	0.06	Trace
Oxidized	.26(MgO)	.78(MnO)	Rare	?

Arsenopyrite is known to exist in some places in the sulphide zone. Some mimetite and pyromorphite have been found in the oxidized zone.

The other principal type of mineralization, important chiefly because of its silver content, bottoms several hundred feet below the general upper limit of the silver-lead-zinc type and extends to the surface. It is associated with many erratic concentrations of high lead, low silver ores and is accompanied commonly by the rare iron silicate ilvaite, which is especially abundant at the lower limit of this mineralization. Quartz is a common gangue mineral; and in unoxidized ore pyrite and pyrrhotite are found, the latter occurring to a much smaller extent than in the silver-lead-zinc type. Analyses of these ores, excluding the high lead, low silver variety, are as follows:

		Percent						
	Ag, oz.	Pb	Zn	Insol.	Fe	S	CaO	MgO
Unoxidized	20.0	3.0	3.0	20.0	23.0	23.0	10.0	3.0
Oxidized	30.0	5.0	1.5	20.0	27.0	4.0	10.0	3.0

This class of ore has been virtually mined out.

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Oxidation extends to a depth of 1,400 feet in several places, but having been governed mainly by late fissures, the depth attained varies considerably.

METHOD OF PROSPECTING AND EXPLORATION

General

Exploration is done from the main operating shafts, nos. 1, 3, and 5. Drifts and crosscuts are run out from these and the necessary raises and winzes driven. Drifts are employed for exploring along main fissure zones, and crosscuts are driven laterally to reach zones east and west of the shafts The shafts themselves give an excellent cross-section of the formation and furnish reliable data for the correlation of horizons and their projection over the entire property.

The main features of a prospecting and exploration campaign are: (1) Detailed geological study of the formation; (2) diamond drilling to supplement the geological study; and (3) the driving of drifts, crosscuts, raises, and winzes, according to the results of the geological study and diamond drilling.

In studying the geology of the property all main shafts first are sectioned thoroughly to determine beds and horizons, whether favorable or unfavorable to deposition of ore. Changes in mineralization are plotted accurately, and in some places channel samples are taken between beds to permit analysis of rock. These data give preliminary information for the projection of horizons between shafts, followed by a detailed study of drifts and crosscuts which makes possible the tracing of beds and favorable horizons, and other major geological features, especially fissure zones. Where shafts are far apart projections may be checked by sectioning intermediate raises and winzes. In new, undeveloped areas where raises are lacking, vertical diamond-drill holes are put in to obtain necessary correlation. The beds, horizons, and other geological features disclosed in the shafts may be projected into known ore bodies and their relative importance observed therein.

In locating a new base for diamond drilling, all the geological data are considered. Holes are pointed for the intersection of the stronger fissures at favorable horizons, experience having shown that such places offer the greatest chances of finding new deposits or extensions of old ore systems.

If a prominent ore manto has been found in a main fissure zone in one part of the property, its horizon may be projected into other fissure systems as a likely objective for drilling.

With accurate geological information, supplemented by diamond drilling, the drifts, crosscuts, and winzes required to reach and explore the ore bodies can be laid out most efficiently. Most of these headings are utilized for ore extraction later.

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Diamond Drilling

Scope.— The diamond drill for many years has been considered a necessary adjunct to the scheme of operation, and its outstanding accomplishments have conclusively proved its great value. The major functions of the diamond-drilling program are as follows:

1. As a prospecting medium.

   a. On occasion it is necessary to do "wildcat" exploration work in remote, unproved sections of the property. In such cases the drill can be used economically to test areas that otherwise could not be explored without undue expense.

   b. Projections of known fissures and horizons may be drilled for ore.

   c. Extensions of ore bodies may be explored.

2. For control of mining.

   Information secured by drilling enables the workings to be laid out with maximum efficiency from the standpoint of haulage and extraction. This in itself is very important in an operation where long, expensive drives are necessary to reach many irregular replacement deposits.

3. To obtain geological data.

   In remote sections of the mine, where little geological information is available, the beds and horizons may be located quickly by vertical up-and-down holes. Rock formations at depth may be studied readily by deep-hole drilling, thereby aiding more intelligent planning of shaft-sinking and exploration programs.

4. To block out ore reserves.

   All of the above-mentioned types of drilling afford information, which is utilized in calculating the volume of ore reserves. Some drilling is done solely for this purpose.

5. For drainage.

   Although drainage seldom is a serious problem at Potosi, drilling occasionally has been used with success to aid rapid unwatering of inaccessible workings.

6. For conduits.

   In places it is convenient to use boreholes for running rubber-covered cable from main levels directly into stopes for connection to electric scraper hoists.

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Organization.- The drill organization consists of a foreman, an assistant, one or more shift bosses, and the operators. The mine geologist lays out the drill bases and holes and does any other technical work requir under direct orders from the superintendent.

Equipment.- Requirements are fully met with the following drills:

1a. One or two EX-size, air-driven drills are employed for holes of 600 to 700 foot maximum and 500 feet average depth at isolated points in the mine where electric drive is not feasible. Ten-foot rods are used, with single- or double-tube core barrels. Black diamonds, ranging in size from 0.75 to 2.0 karats per stone, are used as cutting media. Operators include a runner and helper. (See class 1, table 1.)

b. The same machines may be utilized as one-man units to a normal maximum depth of 350 feet. Black diamonds generally are employed, but at times borts or small natural carbons are substituted. (See class 2, in table 1.)

2a. Two underground EX-size electric drills are used to the same average depth as the class 1 machines. One is equipped with a 7.5-hp. 440-volt, a.c. motor drive and gear transmission, with clutch for variable speed control. A 5-hp., 440-volt, a.c., direct-connected pump is used. This unit has operated successfully for several years; its low operating cost is evidenced by the figures under class 3, table 1.

b. A newly designed drill, which embodies several novel features, has just been placed in service. It is driven by a 5-hp., class BTA, brush-shifting, adjustable-speed, commutator-type, induction motor which operates on 3-phase, 60-cycle, 440-volt alternating current. A 3-hp. direct-connected pump is used with it.

Both machines use single- or double-tube core barrels and 10 foot rods. Borts, diamond scrap, or small natural carbons are employed. The crew comprises a runner and helper.

3. A hydraulic, air-driven, type C drill with A rods is used for special geological work, where deep holes of a maximum depth of 2,500 feet are required. A three-man crew is used. No costs are given for this drill.

4a. Two light, mobile, one-man, air-driven drills are used for prospecting along drifts and in old workings to depths of 200 feet maximum and 150 feet average. Double-tube core barrels and 5 foot rods are used; 3/4-inch core is produced compared with 7/8-inch by the EX drills. Borts ranging 5 to 8 to the karat have been used extensively in the past but now are being supplanted by the more economical small natural carbons. (See class 4, table 1.)

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TABLE 1.- Diamond-drilling department classified costs-Year 1932.

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per foot

	Classes				Averages	
Direct cost	No. 1	No. 2	No. 3	No. 4	1, 2 and 4	All
Operators and helpers	$0.0920	$0.0764	$0.0899	$0.0939	$0.0833	$0.0887
Carbon and borts	.1654	.1190	.0533	.0948	.1329	.1140
Air or power	.0797	.0830	.0094	.0778	.0814	.0643
Carbide	.0020	.0021	.0018	.0034	.0024	.0023
Lubricants	.0044	.0034	.0038	.0035	.0039	.0039
Setting bits	.0347	.0290	.0280	.0493	.0374	.0351
Repairs - machine	.0338	.0237	.0441	.0405	.0330	.0356
Pump repairs	.0025	.0077	.0097	.0064	.0050	.0061
New rods	.0499	.0487	.0412	.0511	.0499	.0479
Supervision	.0843	.0861	.0725	.0853	.0851	.0821
Total direct	$ .5497	$ .4841	$ .3537	$ .5060	$ .5193	$ .4800
Indirect cost						
Tools	$0.0017	$0.0022	$0.0022	$0.0029	$0.0022	$0.0022
Core boxes	.0065	.0059	.0063	.0064	.0063	.0063
Moving drills	.0074	.0050	.0025	.0067	.0065	.0056
Cutting stations	.0169	.0021	.0027	.0019	.0085	.0072
New pumps	.0044	.0071	.0031	.0066	.0065	.0051
Total indirect	$ .0369	$ .0223	$ .0168	$ .0245	$ .0300	$ .0264
Grand total	.5866	.5064	.3705	.5305	.5493	.5064
Footage drilled	15,222	9,275	10,709	9,923	34,420	45,129

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b. Competitive tests are now under way between two Prospector EX electric core drills powered by 3-hp., 440-volt, 3-phase, 60-cycle Westinghouse motors with multispeed drives. One and one-half hp., direct-connected, duplex, double-acting pumps are used. These drills are intended eventually to replace the above described short-hole air-driven ones, have a rated capacity of 250 feet, and are one-man machines. Borts and small natural carbons are used, with such scrap as becomes available from the larger stones.

Relative Advantage of the Several Drill Types.- Until the advent of the newly developed 5-hp. drill with 3-hp. pump the outstanding performer was the 7.5-hp. drill with 5-hp. pump. The former, however, is more compact and efficient and uses less power than the older type. The two short-hole air-driven units have done excellent work in their time but already are being markedly outclassed by the more modern electric Prospector type. The former do not use standard rods or bits, and their maximum range of 200 feet can be extended only by reaming with an EX machine. All factors considered, the limit of maximum drilling efficiency is 400 to 450 feet for all classes of drills except the hydraulic C. The 600-foot-capacity underground air drills used as one-man machines with 5-foot rods (type 1b), the light, short air drills (type 42), and the Prospectors (type 4b) have the advantage of requiring little clearance for setting up, thereby minimizing the expense of station cutting. The type 2 electrics for longer-hole work are decidedly more economical of power than the others, indicating clearly that further application of electric power to core drilling may effect considerable direct savings in compressor operations and reduced power peaks.

Bit Cutting Media.- During the past several years considerable data have been amassed on the relative value of borts, natural carbon, and black carbon ranging in size from scrap to several karats. When carbon prices were high large stones cost $175.00 per karat, and considerable savings were made by substituting borts (5 to 10 per karat). When the market dropped, experiments with small natural carbons showed still further savings, and under current conditions they are preferred. Only such scrap has been found economical to employ as is obtained as a byproduct from large-stone drilling. Other things being equal, the higher priced carbons are utilized least because of the risk involved in losing a bit, together with the higher capital investment represented by expensive stones.

METHODS OF SAMPLING

Underground

Experiments have proved conclusively that at this property the easiest and most efficacious method of sampling for general mining control is taking hand grab samples from mine cars. One from each working place is sampled from the cars at the shaft station, the sample being thrown into a box or can provided for the purpose. Every morning at 7:00 a.m. these samples are collected, tagged with sample and working-place numbers, and

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sent to the grinding room, where they are crushed and quartered; each sample is reduced to an amount that will go into a round tin cap box. The cap boxes are placed in turn in a holder of pipe, slotted in the side for ready manipulation of the boxes, which are held in place at the ends by a spring that takes up any play resulting from not filling the holder. In this convenient and secure container the samples are sent by railway track motor to the general assay office at the mill in San Guillermo.

The mine sampler furnishes the mill daily with the requisite number of assay report sheets, retaining one for telephonic returns early in the afternoon. The assay sheets carry the following information: Date, sample number, contract number, working-place number, and cars mined in each working sampled.

The difficulties inherent to grab-sampling ores carrying appreciable lead values in the form of galena are recognized in Potosi sampling practice. This is secondarily true of silver and zinc values. Daily assay determinations are made for silver, lead, and zinc and the results adjusted by predetermined factors for the three metals. The factors are obtained by comparing the geometric average of all assay returns during the month with the average monthly mill heads, which are obtained from weightometer and automatic sampling at San Guillermo. For example, comparisons may show the mine-car samples to be 12 percent high on silver, 20 percent high on lead, and 5 percent low on zinc. Beginning the following month the mill assayer applies these corrections, and in consequence differences will be very slight at the end of the second month. Thus errors common to grab sampling are substantially discounted in making the mine-assay return.

Channel sampling is used for special purposes only. Occasionally it is useful in a stope for tracing an otherwise obscure or unrecognizable fissure, as may well be the case in a limestone formation. The plotted values usually indicate segregation of certain metals where the fissure goes through the body. Channel samples also are applied occasionally as a phase of the detailed study of low-grade stopes for selective mining. In this instance composite assays are made of groups of samples to determine such other constituents as sulphide, lime, manganese, iron, magnesium, copper, and insoluble.

The grade of average mine-run ore may be estimated readily by inspection, as the operator's experience permits this close appraisal of values. On the other hand, carefully compiled records of stope assays are necessary for geological purposes, estimation of ore reserves, and proper preservation of stope histories. Monthly, the assays from each working stope and producing face are averaged geometrically and the total production and grade plotted on 1:2500-scale level maps, using a distinctive color for each month. This record includes the contract number, tons of ore produced, and assay values in terms of silver, lead, and zinc.

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Diamond-Drill Sampling

Use of the double-tube core barrel makes possible a core recovery as high as 95 percent, if desired. It is common practice to obtain high core recovery when drilling in sections where little or no prior information is on record, but in known ground the runner is instructed to grind core until he nears his objective. This procedure is ordered for a specified hole only and in no case is left to the judgment of the runner.

When ore is cut the hole is cleaned out before drilling farther. Rods are pulled every 5 to 10 feet, according to the length of the core barrel. Core samples are assembled in portable cases and taken to the diamond-drill core room, where they are classified by the geological department and filed away as permanent records. Core samples of ore are split lengthwise, one half being filed and the remainder analyzed. No sludge samples are taken, as Potosi sulphides core readily and give the required physical and chemical data.

ESTIMATION OF TONNAGES

Tonnage estimates are based, as far as possible, on ore limits exposed in workings and diamond-drill holes. The formula for the volume of the frustrum of a pyramid is used for calculating the tonnage of irregular blocks of ore:

$$V = \frac{(a + \sqrt{ab} + b)\ T,}{3F}$$

in which V = short tons,
a = top area, square feet,
b = base area, square feet,
T = vertical height of block, feet,
F = factor, cubic feet per short ton of ore.

Deductions for inclusions of waste up to 20 percent are made in accordance with conditions.

Estimating ore reserves in a limestone replacement deposit such El Potosi is difficult at best. A small, unexplored streak in the wall of an abandoned stope may disclose new possibilities; and, conversely, a strong ore trend or section may overnight reveal large horses of waste or noncommercial ore. Furthermore, development work in such deposits is less systematic and less useful in estimating ore reserves than in deposits of regular form. In many cases standard methods of calculating volumes are not feasible, and estimates are based on past history of the deposit and an experienced view of conditions.

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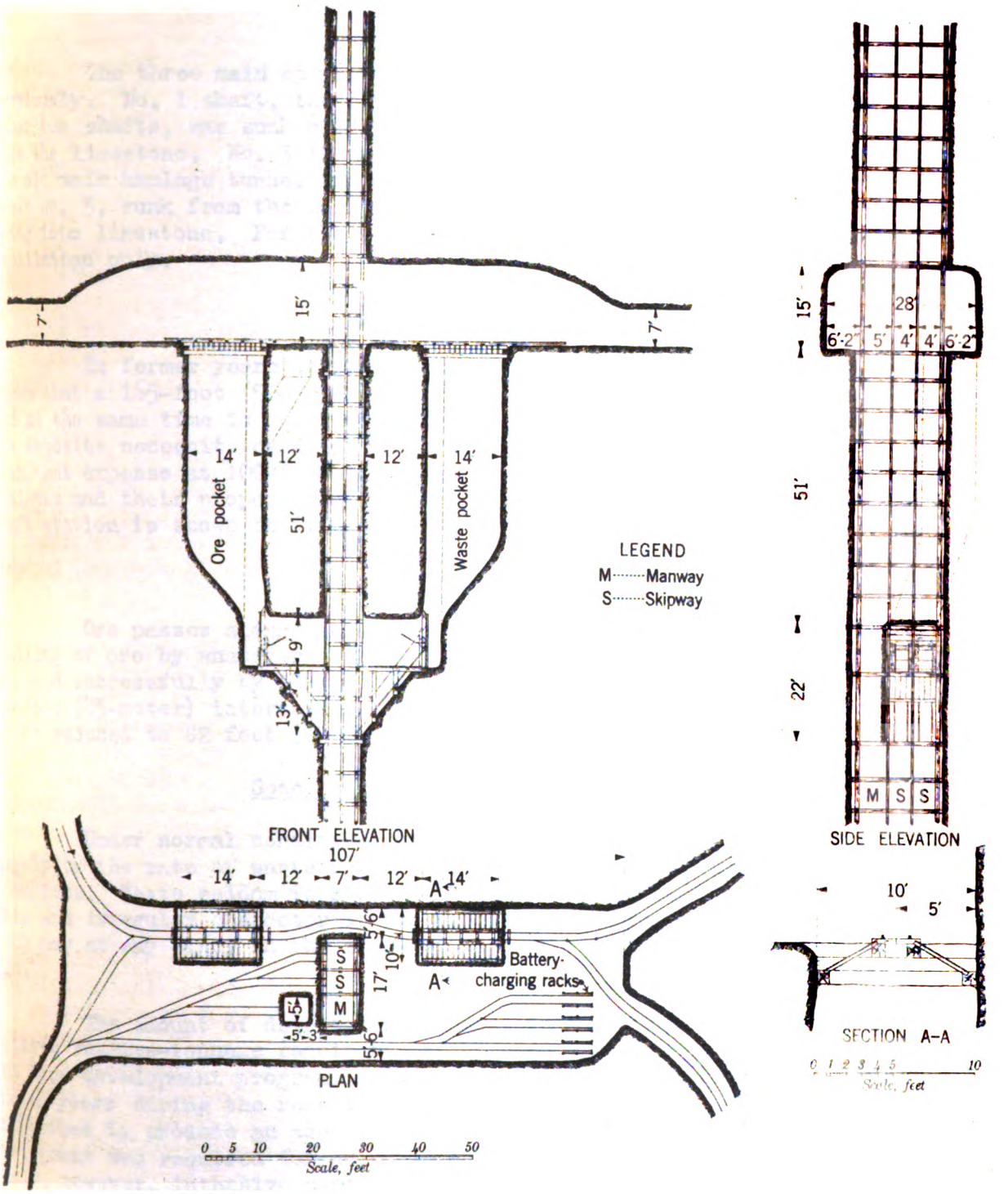

Figure 3.—Standard shaft-station layout.

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METHODS OF MINING AND DEVELOPMENT

Shafts

The three main operating shafts are nos. 1, 3, and 5, mentioned previously. No. 1 shaft, the collar of which is 65 feet above those of the other two shafts, was sunk 620 feet through volcanic capping before going into the limestone. No. 3 is an interior shaft sunk at the patio level from the old main haulage tunnel and enters limestone at a depth of 225 feet. Shaft no. 5, sunk from the main patio level, cuts 565 feet of capping before going into limestone. For the past two years no. 3 shaft has been used for ventilation only.

Level Intervals

In former years level intervals were 100 feet, but experience has shown that a 165-foot (50-meter) distance serves major requirements better and at the same time is more economical and practical. The irregularity of the deposits necessitates driving long headings to reach objectives, and the resultant expense at 100-foot intervals is prohibitive. The use of intermediates and their proper location is discussed under Stoping. A standard shaft station is shown in figure 3.

Ore Passes

Ore passes along haulage levels are spaced to require a minimum handling of ore by whatever stoping method is used. If a block of ground can be mined successfully by the glory-hole method, followed by scraping, a 150-foot (45-meter) interval is satisfactory. In other cases this distance may be reduced to 82 feet (25-meters).

Speed and Amount of Development

Under normal conditions the speed of development work is governed largely by the rate of waste removal both at the face and from shaft pockets to surface. Waste seldom is stored or gobbed underground owing to the erratic and irregular character of the deposits, the worked-out sections of which may at any time, in the light of new information, require reinvestigation.

The amount of development depends on many factors, principally: Positive reserve-tonnage requirements, rate of extraction, percentage of return from development program, and success of the drilling program. For several years during the rehabilitation era of the property, when it was being conditioned to produce an adequate tonnage of milling ore, about 1 foot of development was required for every 10 or 12 tons of ore extracted. More recently, however, intensive concentration upon the geology, with close supervision of all diamond drilling, has resulted in a ratio approximating 1 foot of development for every 30 tons mined.

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Shaft-Sinking

The sizes of the normally active shafts are given in the following table:

Shaft no.	Total depth, feet	Depth of sump, feet	Dimensions, feet and inches			Size of timbering, inches 1/
			Over-all	Compartments		
				Hoisting	Manway	
1	2,777	149	8 by 17	4 by 5	5 by 5	10 by 10 6 by 10
3	2,235	13	7-6 by 16-6	4 by 5	5 by 5	8 by 8 6 by 8
5	2,587	35	7-6 by 15	3-8 by 5	4 by 5	8 by 8 6 by 8

1/ Standard shaft sets of creosoted Texas pine.

A cycle is completed each day in shaft sinking. In a typical sinking operation on the day shift, a crew of 3 machinemen and 1 hoistman drills a round, hoists the tools and equipment, and blasts. On the afternoon shift a crew of 4 muckers and 1 hoistman loads out most of the broken material from the round. On the night shift the same-size crew finishes mucking out the round and lowers plumb lines and the tools for the succeeding shift.

Figure 4 shows the progressive stages of sinking operations. The shaft is sunk first in the manway compartment far enough below the bottom of the sump in the main hoisting compartment to allow for a protective pillar (pentice) 15 feet thick, following which sinking continues in full cross section. As each successive level interval is reached short headings are driven out (see level 16, fig. 4) before station cutting. Meanwhile, 60 fee below the level similar headings are driven and raises started for pocket excavations, following which the bottoms of the headings are taken up and small temporary waste pockets formed for catching the waste blasted out of the pocket raises. These raises are then driven to the level station and reamed out to their required size and shape (see levels 14 and 15, fig. 4). Only one hoist set-up is required for every three levels, provided the hoist has enough power, speed, and capacity.

A V - cut round is employed, using 32 to 35 holes loaded with 160 sticks of 1-1/8-by 8-inch dynamite and detonated with no. 6 electric blasting caps (fig. 5). The latter are fired from the power line (protected by a safety switch box) or by battery. The round breaks approximately 4.5 fee The ground is easy to drill but is very tenacious and springy and therefore hard to blast. The broken rock is handled through the manway compartment b an electric hoist; the size of the hoist motor ranges from 20 to 75 hp., a cording to the hoisting depth. Two sinking buckets are employed, one of which is filled while the other is hoisted. At the time of dumping workmen are protected from falling rocks by a safety door. The hoistman, from his

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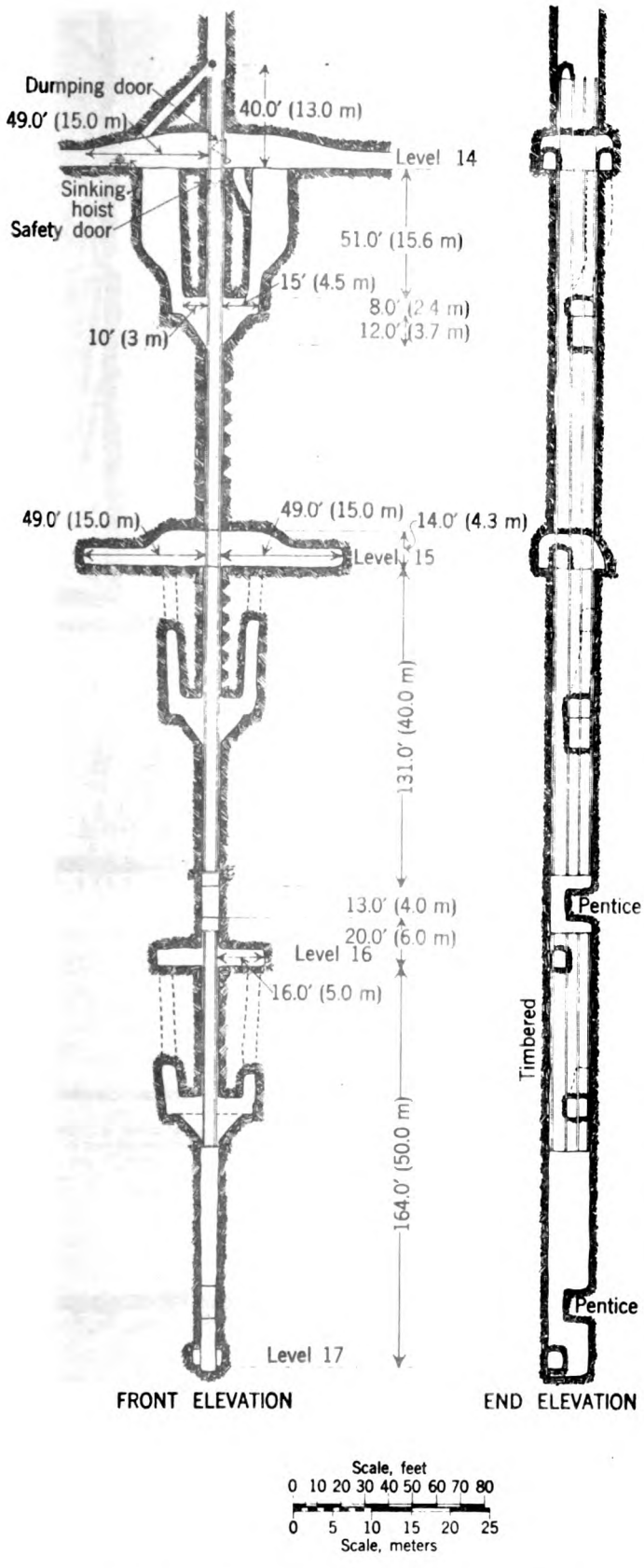

Figure 4.—Progressive stages of sinking operations at Potosi No. 1 shaft.

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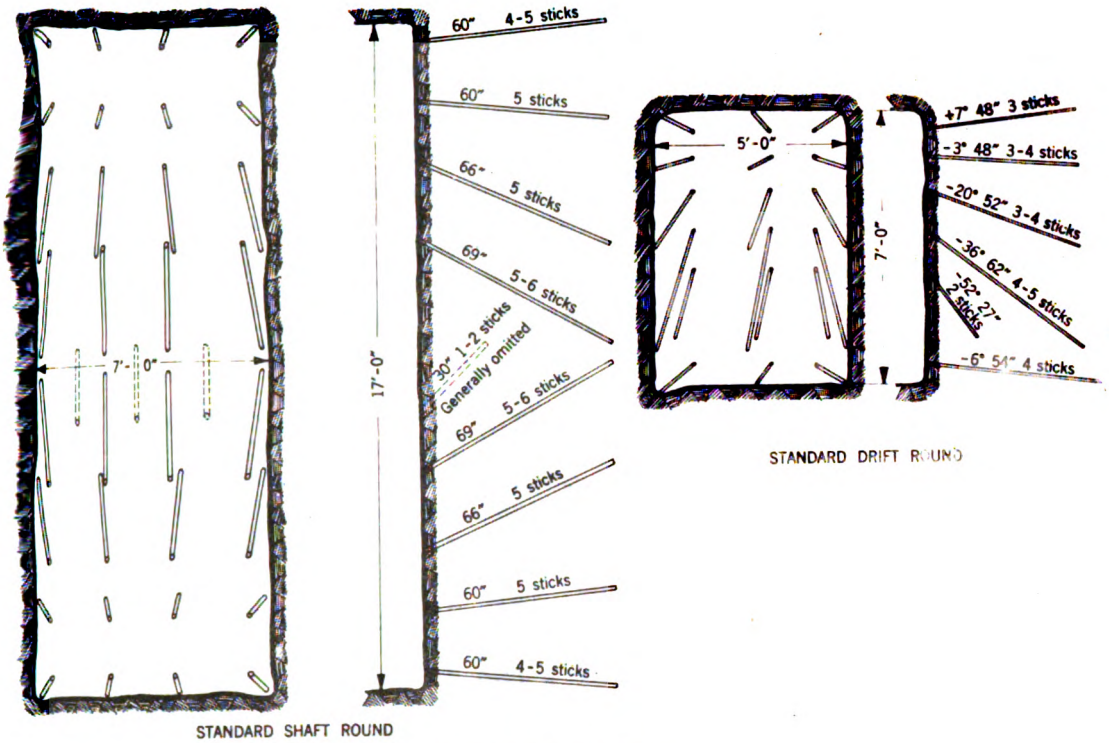

Figure 5.—Shaft and drift rounds.

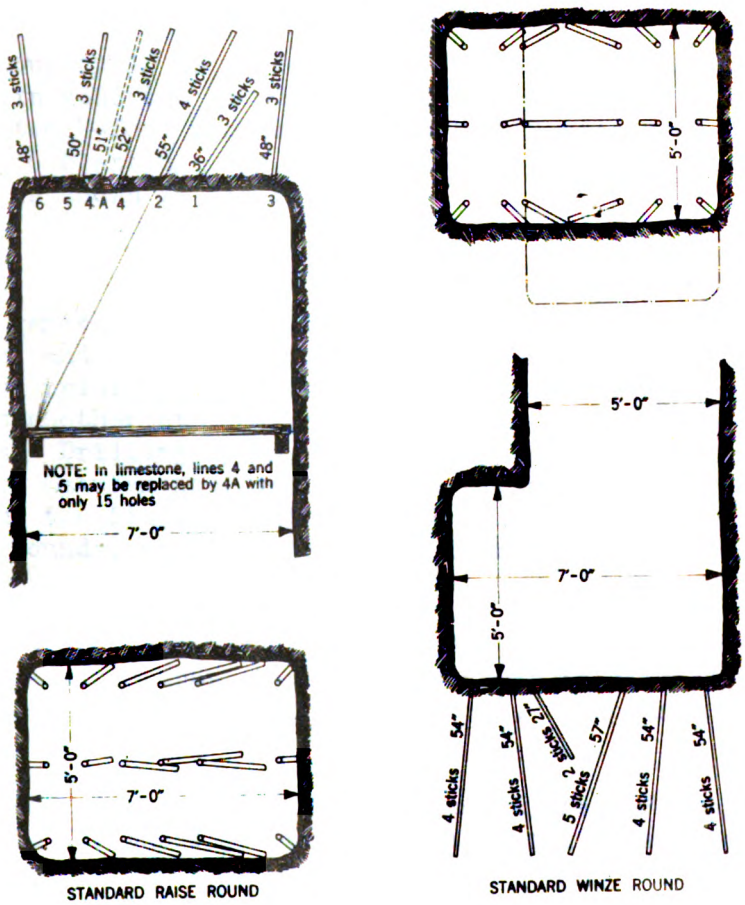

Figure 6.—Raise and winze rounds.

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position at the hoist, places the dumping door in position and empties the bucket by means of a system of levers and counterweights. The waste discharges directly into a main shaft pocket, from which it is handled by the main hoist. The sinking bucket runs on wire-rope guides and is equipped with a crosshead.

Drifting

A standard drift round in limestone consists of 18 holes using a toe cut (fig. 5, B). The drill crew comprises 1 machineman, 1 helper, and 2 mucker-trammers. Normally, 65 sticks of 1-1/8 by 8-inch dynamite are used, with 18 caps and 100 feet of fuse. A round breaks 3 to 4 feet and makes 12 cars of muck in a 5-by 7-foot heading. A round is drilled and blasted, and the rock from the previous round is mucked out in one shift.

Winzes

In a winze 3 machinemen and 6 muckers drill, blast, and muck 1-1/2 four-foot rounds in 24 hours. When not drilling the machinemen run the small air or electric sinking hoist. A winze round is shown in figure 6.

Raising

A 24-hour cycle in raise work requires 3 machinemen, 3 helpers, and 3 muckers, who, in addition to putting in the necessary stulls and staging, drill and blast two 4-foot rounds. Normally 114 sticks of 1-1/8 by 8-inch dynamite, 36 no. 6 blasting caps, and 250 feet of fuse are required. A typical raise round is shown in figure 6.

Classification of Rock Drills

Three general types of rock drills are in use - mounted light drifters, jackhammers, and stopers. The makes now used were chosen after a 6-month competitive trial in which the contending drills were changed from one working place to another at regular intervals to subject them to average operating conditions. Drilling speed, air consumption, general adaptability to the work, repair, maintenance, and over-all efficiency were all contributory factors in making the final selection. The drifters weigh 140 pounds, the jackhammers 57 pounds, and the stopers 90 pounds. All machines are operated wet.

Drifters are used for driving headings and to some extent for ore breaking in the larger stopes. Jackhammers serve in sinking shafts and winzes but principally in underhand stoping operations. Stopers are utilized in raise work and occasionally in stoping.

All machines are equipped with line oilers. Foremen and shift bosses are responsible for their proper lubrication and make a written report on

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them every 15 days. The following features are included:

1. Contractors' name and number.

2. Machine type and serial number.

3. Whether machine is operating or acting as extra.

4. Whether water and lubrication are functioning properly.

5. Condition of accessories such as air hose, line oilers, water tank, water line, air lines, and intermediate valves.

Two-braid, 3/8-inch water hose and three-braid, 3/4-inch air hose have superseded ply hose following 2 years' testing, which proved lower ultimate cost of the former.

Rock Drill Records

Contractors are required to render a daily report of the hours worked by each individual drill under their charge, so that the "shifts in service" may be tabulated. When the machines are sent to the central repair shop, the mechanical department makes a record of repairs or renewal of parts, the cost, and the life, in shifts, of the particular part repaired or replaced. Notation is also made whether the part is factory or locally made. Every 6 months a repair-part analysis is made for each machine.

All parts showing a life less than the established normal are checked to determine whether they are defective or have failed due to carelessness of the operator. Defective parts are sent to the manufacturer for examination and presentation of claims based on the life of the defective parts as compared with the normal. This has been of benefit to both Company and manufacturer in ironing out "kinks" and minimizing wear.

The compressor output is metered, and the cost of compressed air is proportioned to each machine according to the average consumption for each type (as established by periodic tests) and the hours of service. For comparison it has been found that "cost per machine-shift" is sufficiently accurate and much more practical than the former basis of "cost per foot drilled."

Monthly tabulations of the average total cost of each machine are assembled into contract groups; over a period of time these records indicate the relative care or abuse to which equipment is subjected by each individual contractor. Irregularities thus may be localized and corrected.

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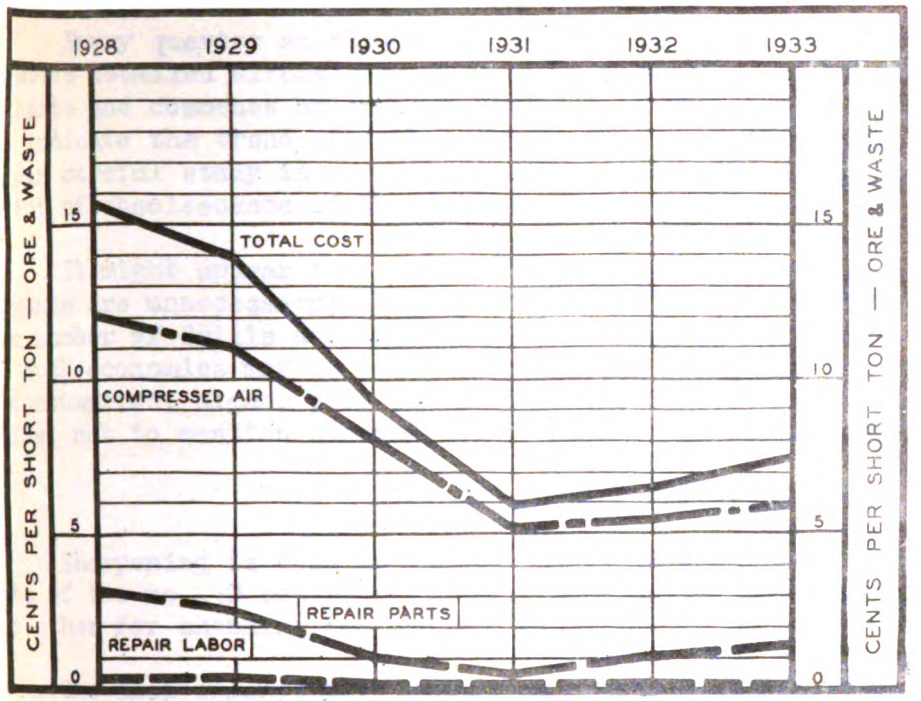

Figure 7.— Air-drill expense.

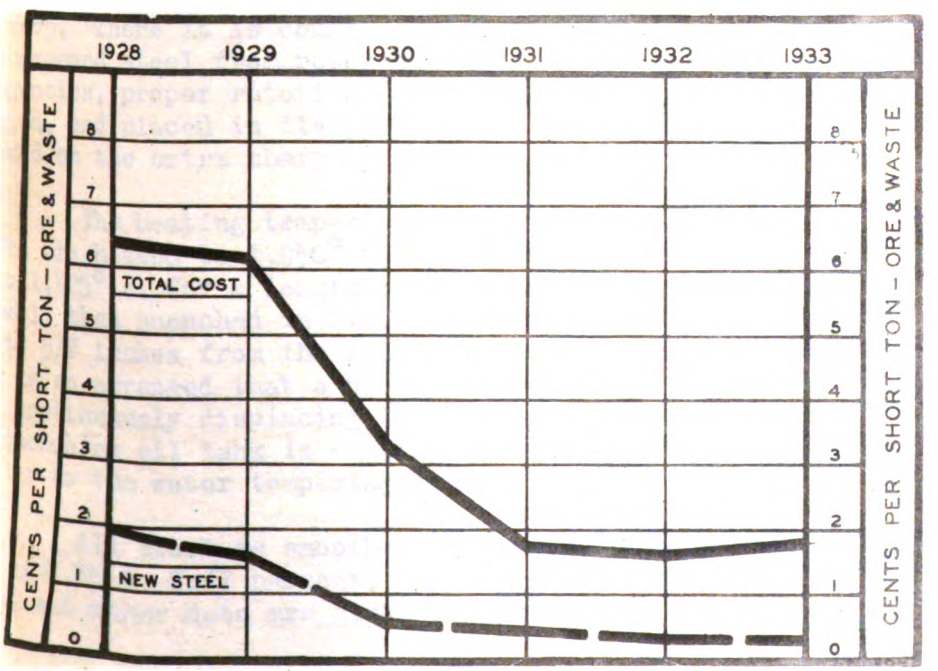

Figure 8.— Steel-sharpening costs.

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Every quarter an air-drill analysis report is compiled, showing comparative detailed direct costs for several previous quarters, together with charts and comments on salient features of the data (see fig. 7). The charts indicate the trend of each major component of the total direct cost, and their careful study in comparison with previous trends helps to determine the point of obsolescence of the drills.

It might appear that such refinements in the study of air-drill performance are unnecessary; but it has been proved at this property, where a large number of drills are operating, that worthwhile economies are effected thus. Such economies may be both direct and indirect and include reduced power consumption, higher drilling efficiency, lower upkeep, and fewer drilling units, not to mention numerous other advantages of secondary importance.

Steel-Sharpening Practice

Sharpening is done on the day shift at a central shop. Equipment consists of two no. 50 sharpeners, one of which is used only for sharpening and the other for shanking new and old steel.

The shop organization comprises 6 men—2 operators and their helpers, who run the machines; a helper who straightens and clears bent or plugged steel and puts the old steel in shape; and, 1 tempering man, who is responsible for all tempering work.

Underground the dull steel is assembled in small steel-cars from the various working places and hoisted to the surface, where it is put into a large car at no. 1 shaft for delivery every 24 hours to the central sharpening shop. There it is counted and immediately replaced by an equal amount of sharpened steel from reserve stock, which is large enough to provide for emergencies, proper rotation, and rest. The dull steel is straightened, unplugged, and placed in its proper rack. Bad shanks and broken pieces are reshanked on the extra sharpening machine.

The heating temperature for forging drill bits is 1,850° F. Drill shanks are heated to 1,950° to 2,000° F. In tempering, shanks are heated to about 1,425° F. for a length of 6 inches, so that the lugs may be tempered as well, then quenched in oil. Drill bits are heated to about 1,425° F. for 1/2 to 5/8 inches from the end and then quenched in water. The tempering tank is so arranged that a plentiful supply of cool water enters at the bottom, continuously displacing the hot water which discharges from the top. The quenching oil tank is enclosed in a larger tank filled with running water similar to the water tempering tank.

All steel is smooth-bore, 1-inch, hollow round, with a carbon content of 0.75 to 0.87 percent. Lugged shanks and crossbits are used. Lengths, gages, and other data are given below:

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	Length, feet - inches	Gage, inches
Starter	2 - 6	1-7/8
2d steel	4 - 6	1-3/4
3d steel	5 - 6	1-5/8
4th steel	7 - 0	1-1/2
5th steel	8 - 6	1-3/8
6th steel	10 - 0	1-1/4

Wing thickness, inch	5/8
Angle of reaming edge, degrees	5
Wing taper, degrees	14
Angle of cutting edge, degrees	90
Diameter of hole, inch	5/16 to 3/8
Diameter of bit at bottom of groove, inch	1

The average consumption of steel is 0.0392 pound per ton of ore and waste mined. Steel-sharpening costs for several years are charted in figure 8.

Explosives

Thirty-five percent ammonia gelatin dynamite is used at the mine. The cartridges are chiefly 1-1/8 inches in diameter and 8 inches in length with the name of the company stamped on each cartridge. Some 1-inch-diameter powder also is used. An endeavor is made to use stemming to obtain maximum blasting efficiency. Dynamite is stored on the surface in a magazine conforming to accepted safety regulations as to ventilation, lighting, etc. Explosives are issued to contractors once every 24 hours. There are no underground magazines.

No. 6 blasting caps are used with a standard brand of fuse in all work except sinking in winzes and shafts, where an improved no. 6 electric blasting cap is employed. Caps are not transported in the same cage with dynamite. They are also stored at some distance from the powder magazine on the surface.

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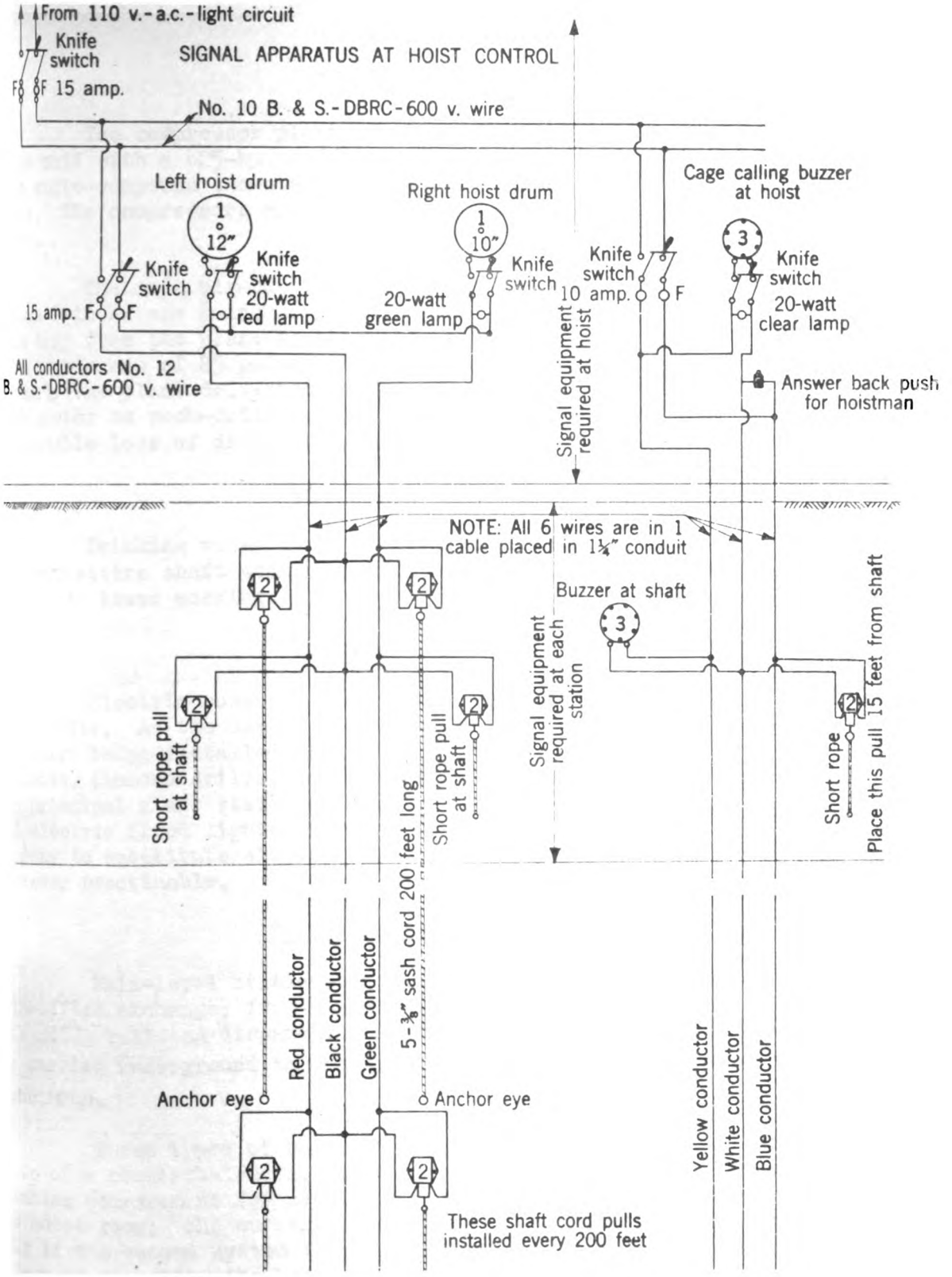

Figure 9.—Electric shaft-signaling system: 1, Single-stroke bells; 2, mine-signal pull switches; 3, heavy-duty, 110-volt mine buzzers; 4, 3-wire, 30-ampere, 250-volt conduit; 5, ⅜-inch steel-center sash cord.

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Compressed-Air Plant

The compressor plant consists of 1 Nordberg 4,000-cubic-foot two-stage unit with a 625-hp. synchronous motor and 3 Sullivan 2,210-cubic-foot twin angle-compound two-stage compressors driven by a 400-hp. synchronous motor. The compressors are equipped with air filters to protect the air intakes.

The main air-distributing system from the compressor plant consists of one 8-inch, one 6-inch, and two 4-inch lines. There is a very slight pressure drop from the plant to the main workings, and after considerable experiment a pressure of 85 pounds at the compressor has been adopted as standard. Formerly the plant delivered air at 100 pounds; but the lower pressure is much easier on rock-drill equipment, resulting in lower upkeep costs without appreciable loss of drill-operating efficiency.

Underground Water Supply

Drinking water is piped from the domestic water supply to outlets on every active shaft station. Water for drilling is obtained from unwatering of the lower workings.

Underground Power Supply

Electric power is available underground for various uses at 110 to 2,200 volts. As the use of electric power becomes more general, main trunk lines are being installed along the principal haulage drifts to supply scraper units, diamond drills, small blowers, hoists, charging stations, and pumps. The principal shaft stations and measuring pockets are lighted by electricity, and electric flood lights are used in scraper stopes. It is the policy of the company to substitute electric power for the more expensive compressed air wherever practicable.

Telephone and Signal Systems

Main-level stations in each shaft are connected by telephone to the main-office exchange, from which connection can be made with the general office, mill, railroad dispatching system, and long distance. Telephone lines are carried underground in conduit; telephone instruments are of the Mine-a-phone type.

Three types of bell signals are used in the shafts. The first consists of a counterbalanced, light-steel cable suspended in one corner of each hoisting compartment for closing a switch on surface, which rings a bell in the hoist room; the current employed is from a wet battery. This is for use only if the second system is out of order. The latter is employed by cage riders in signaling the hoist engineer. A braided cord with steel center is suspended in one corner of each hoisting compartment in 200-foot segments, each connected to a type-FA spring contact, which is adjusted against the weight of the cord and actuates a bell signal and colored light in the hoist room (fig. 9). The third system comprises pull switches and buzzers at every

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shaft station and pocket and a buzzer, light, and answering pushbutton switch at the hoist. It is used for calling the cage. The last two signal systems are operated by 110-volt alternating current.

STOPING

The stoping method used at Potosi depends in large measure on the favorable strength of the country rock, which obviates the necessity of timber support or filling. The propensity of the sulphide ore to oxidize precludes employing methods that would allow large amounts of broken ore to remain long in the stopes. Furthermore, the irregularity of the deposits necessitates investigation of all patches, seams, and stringers exposed in the natural course of mining.

Mining comprises four steps: (1) Exploration and development in which the ore body is blocked out by drifting, raising, crosscutting, and diamond drilling; (2) underground glory-hole mining (underhand stoping) in which the ore is benched out around centrally located ore passes; (3) mechanical slushing or scraping, which is substituted for glory-holing when the stope face has reached the angle of repose or is too far back from the chute for handling by gravity; and (4) the final clean-up in which experienced men follow out the patches, seams, and stringers of ore exposed in the natural course of mining. The importance of this clean-up cannot be overestimated; emphasis thereon often is responsible for outstanding ore-reserve increments.

Whether all or part of these methods are employed depends on the extent of the deposit being worked. All are necessary, in the order named, in ore bodies of large lateral and vertical extent, that is, the chimney and ultrathick manto systems. In a small manto stope prospecting and development usually are followed by simple stoping, in which the ore is mainly shoveled by hand into cars or directly into a chute. This corresponds to clean-up work as employed in large stopes. In other cases development and prospecting open up mantos that are not thick enough to permit glory-hole mining but are ideal in size and shape for mechanical mucking, followed in its final stage by clean-up work.

In developing a chimney-type ore body crosscuts and drifts are driven through it on two levels to define better the general limits obtained by preliminary prospecting (usually diamond drilling). Then from the higher of these two levels raises are driven through to the limestone, while at the same time the sill floor is mined out to the waste contact. Should the chimney's vertical extent prove too great, headings again are run in from a higher level; or, if more feasible, connection is made from contiguous workings. Once the limestone back is cut work is begun on stripping the ore against the contact, along with initial benching around the glory hole. An extraction raise of the H type (fig. 10) provides a manway between the haulage level and the bulldozing chamber and at the same time allows natural ventilation between the level and the stope. Ore in the chute never is permitted to build up so far above the intermediate drift as to cut off the

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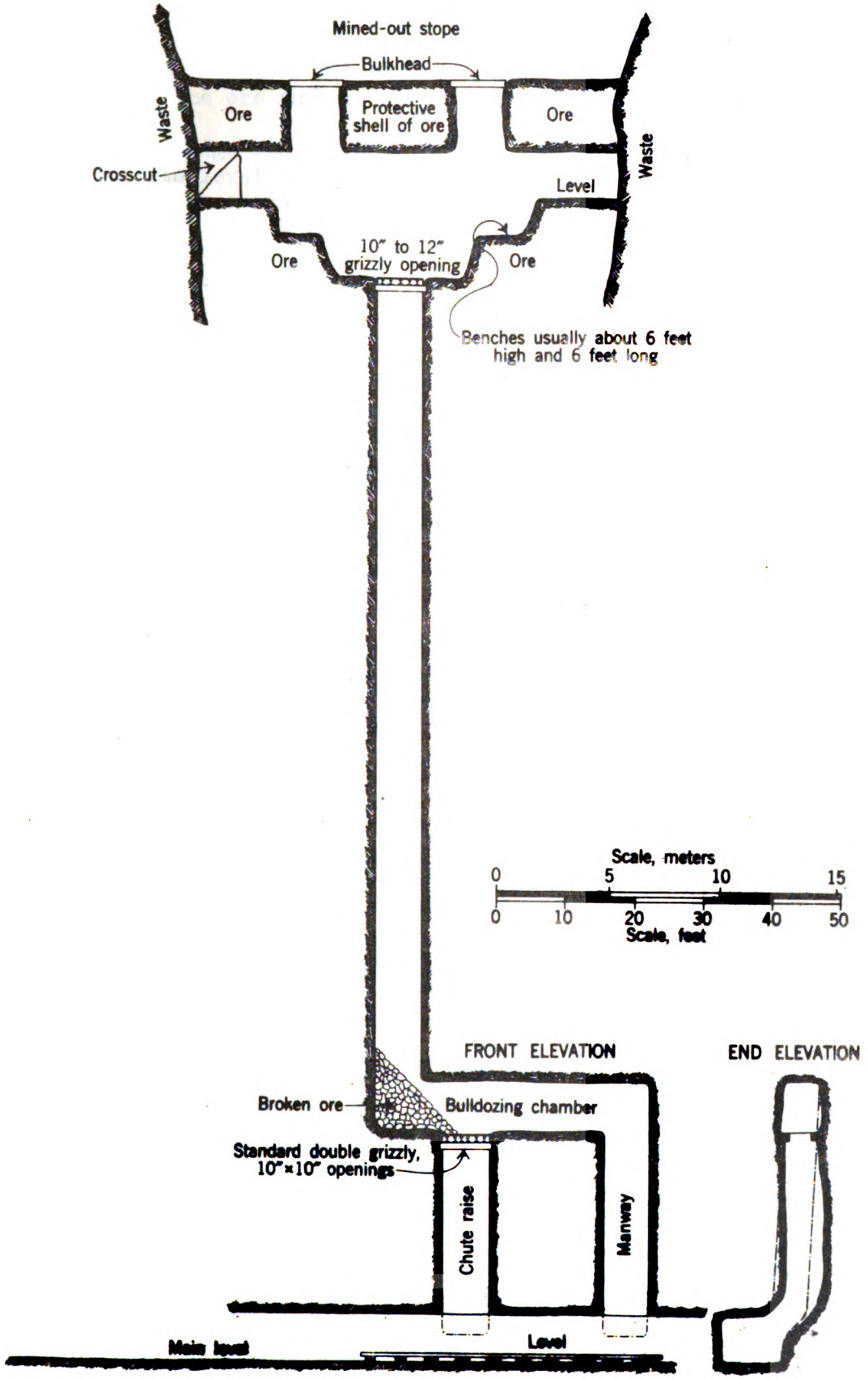

Figure 10.—Standard "H" raise and glory-hole stope.

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natural draft. The distance between the grizzly and the main level provides enough storage in the ore chute.

In an average chimney the vertical mining range for a given block usually is from one level to the next (165 feet), but in a chimney of unusually large cross-section the first block goes down to an intermediate about halfway to the next level. A 10-to 15-foot shell of ore always is left directly above the level or intermediate as a protection until the section below it is finished (see fig. 10).

In glory-holing, benches are begun around the connecting winze by drilling 6-to 8-foot inclined holes 3 feet apart and about 3 feet back from the winze edge; the first cut requires two concentric circles of holes. While blasting the first 2 or 3 steps or benches it is very difficult to use a permanent grizzly because of its close proximity to the work, but old timber and rail are laid across the chute opening temporarily until enough advance has been made to allow replacement with a permanent installation. The latter consists of old rails fastened 10 to 12 inches apart on wooden stringers or sills. It will be noted (fig. 10) that the ore makes its own incline at the bottom of the raise and then falls over the grizzly in the bulldozing chamber.

Access to glory-hole benches or manto stopes is gained directly from the level above or occasionally from special manway raises connected into them. Roadways are provided down onto the benches by wire-rope ladders on steep walls or pathways with cable handholds along the benches themselves. Wherever the men are exposed to any danger from slipping down the glory hole or bench they are safeguarded by a rope around their waists.

In blocking out a large manto its direction and extent are determined first, then an extraction heading is run under it in the direction of its long axis. H raises are then driven from the level to the back of the ore. If an ore body consists of a number of mantos lying in an ore zone H raises are driven through the manto series, and the minimum possible number of intermediates is driven from them to effect economical extraction. In many instances the relatively thin limestone beds separating the ore mantos can be stripped by slushing.

Drilling and Blasting

Methods of drilling and blasting in stopes depend upon conditions in the individual workings. Glory-hole mining—that is, underhand stoping—employs down holes, which bench out the ore around the ore pass. As shown in figure 10 the benches usually are about 6 feet high and 6 feet wide. Drilling and blasting long holes (10 to 12 feet) resulted in such coarse ore that scraping was hampered unduly by the many large slabs. When mining outward on the side of a deposit of manto form either mounted drifters or jackhammers are provided. If the ore is "patchy" and erratic, it is easier to use a jackhammer than a mounted drifter. No. 6 blasting caps are normally used, but on some special jobs no. 6 electric detonators are preferred.

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Stope Support

The character of the formation simplifies stope support, both in ore and limestone. Timbering rarely is required; in large mantos extensive exposure of the stope back occasionally may necessitate timber cribbing. As a rule, however, irregular pillars are left where dictated by the operator's judgment. In stopes having high backs, especially in chimney deposits, the preliminary work includes careful arching of the backs to give proper roof support. As the stopes are dry, it is possible in many cases to arch the backs, in limestone, for widths of 100 to 150 feet, and lengths of 200 to 300 feet, then to mine downward for several hundred feet, without danger.

Handling Ore in Stopes

Ore is handled in the stopes and auxiliary workings by hand-shoveling, by gravity, or by mechanical mucking. Hand-shoveling is used in the preliminary stages of blocking out and in cleaning up. Every opportunity is taken to move the ore by gravity from the face to the chutes. Wheelbarrows and small mine cars are required occasionally for short periods in intermediates, pending preparation of the workings for more efficient handling methods.

Slushing.- Mechanical mucking or slushing is used largely in the principal stopes, especially those of manto type, where the ore is scraped directly into chutes. Its introduction has done much to lower costs in many workings which otherwise could be mined only by hand at prohibitive cost. Sullivan double-and triple-drum hoists are used, including four 10-hp., one 6.5-hp., and one 7.5-hp., double-drum, and seven 15-hp., triple-drum hoists driven by 440-volt a.c. motors. The latter are much preferred because of their flexibility. With them it is possible to muck over a large area from one set-up, whereas with the two-drum machines it is constantly necessary to change the location of sheave blocks. A typical scraper set-up in a manto stope is shown in figure 11. Sheave wheels and scrapers (fig. 12), typical of those used elsewhere in mining, are made in the company's own shops.

The normal slushing range in stopes is 100 to 125 feet, but this casionally is increased for a short time to 150 feet. Some loss in efficiency is noted in work at the latter distance, unless conditions warrant using a higher-speed scraper motor.

Recently slushing has been applied successfully to loading cars from the stope floor. A timbered slide is constructed somewhat similar in design to the gravel-loading platforms used by road contractors. A 15-hp. triple-drum hoist is mounted on the back end of the platform at the far side of the opening above the mine car. With three-drum control the operator can scrape the ore readily to a central pile in front of the bottom of the slide until he has accumulated enough to continue up the slide and discharge the load through the open trap. Such an installation is warranted where a considerable tonnage of ore can be handled from a central set-up.

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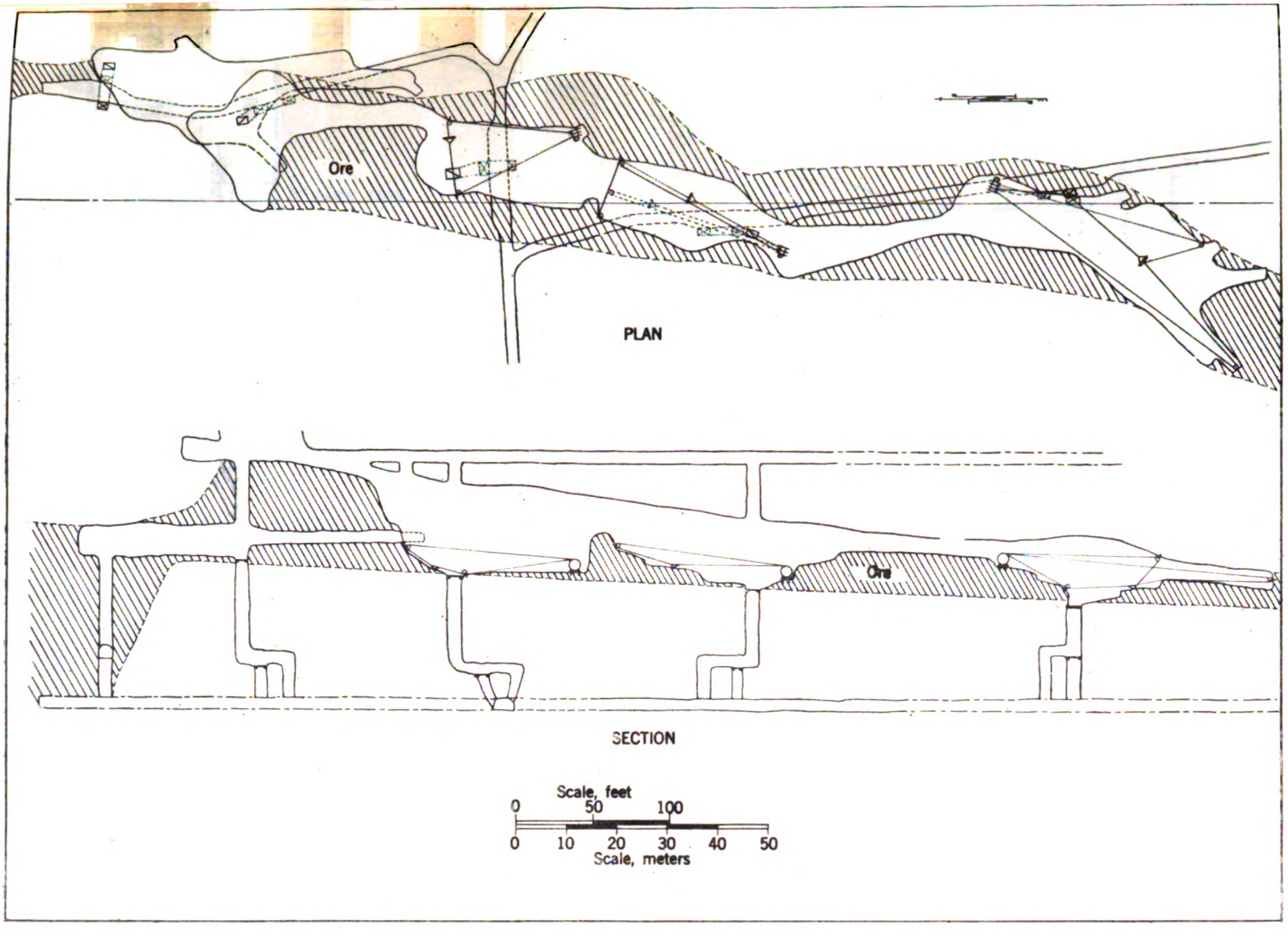

Figure 11.—Slusher layouts in stope.

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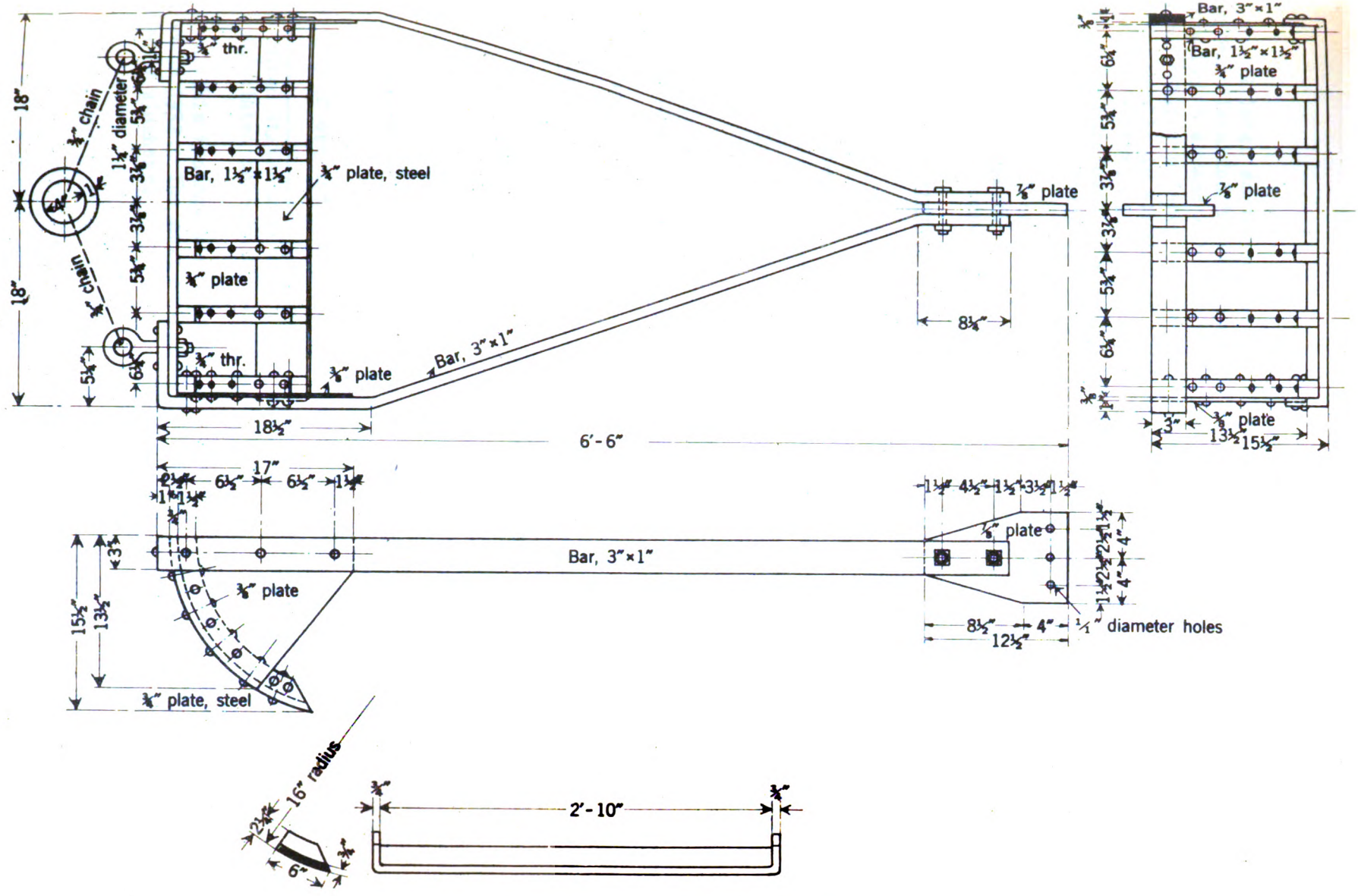

Figure 12.—Scraper.

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A large saving has been effected by using cable strands salvaged from discarded hoisting rope. One and one-eighth- and one and one-quarter-inch 6 strand cable is unstranded for this purpose, the only cost involved being the labor of unstranding and installation. The rope causes a little trouble in winding at first until the strand straightens out, then comparatively little is experienced. Its life naturally is less than that of new cable purchased especially for the purpose, but the saving far outweighs the slight disadvantages.

Mechanical scraping, when applicable, can readily save $0.10 to $0.30 (United States currency) per short ton of ore moved, through the gain in tons handled per man-shift.

Underground scraper costs; year 1932

Item	Cost per short ton[1/]
Making and repairing scrapers	$0.0054
Power lines	.0004
Hoist installations	.0056
Hoist repairs	.0167
New cable and cable repairs	.0038
Electric power	.0056
Lubrication	.0014
Total	$ 0.0388
Short tons of ore and waste handled in 1932	107,420

1/ In United States currency. Cost of labor and Mexican supplies calculated on exchange rate of 2 to 1.

This tonnage represented 24 percent of the total amount of milling ore in 1932. The next year 154,754 tons (34 percent of the total) was handled by slushers.

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Thus far mechanical scraper loaders have been used underground in stoping operations only, but a slide loader was successfully employed in driving the 12-by 12-foot railroad tunnel on the patio level to the new ore-bin site at no. 1 shaft.

Sorting Waste

In chimney-type ore bodies little waste rock is encountered in the natural course of mining until stoping nears the edge of the deposit. Limestone is then likely to be found in places as isolated horses, scattered fragments, or boulders (resulting from imperfect replacement of the limestone) or as mantos of variable thickness. Manto deposits are more likely to have measurable waste inclusions. If the amount is small it is thrown to one side in the stope, but if a sizable waste manto or horse is encountered it may be necessary to discontinue ore breaking and spend several shifts on waste material. As observed heretofore, often it is feasible to handle considerable quantities of waste by electric slusher hoist. Some additional sorting is done on the bulldozing-chamber grizzlies. The final sorting, however, is done on the picking belt after the ore has passed through th crusher on the surface.

The amount of sorting done is governed largely by the economics of each case. To effect the lowest ultimate stoping cost large tonnages must be moved, and close sorting may so hamper the program as to defeat its purpose.

Grizzlies

The primary blasting breaks most of the ore small enough to pass through 10-inch grizzly openings. Large boulders are drilled and blasted and others are broken by hammering; most secondary breaking is done in the stope but some in the intermediate of the H raises. Grizzlies are placed over all ore raises in the stopes, not only as a safety measure but also to prevent large boulders going into the chute and blocking it. When a stope is worked nearly to a main haulage level it may be unnecessary to have any grizzly other than the one over the ore pass. In a high raise it is always advisable to have a second grizzly in the H raise, as shown by figure 10. The larger pieces fall to one side in the intermediate, where they are broken and thrown back into the chute. The upper grizzlies are of parallel rails 10 to 12 inches apart; the lower ones are of crossed rails, with 10-inch-square openin

Tons per Man-Shift

Figure 13 is a chart showing the tons of ore per man-shift by years from 1928 to 1933 (1) for the mining department and (2) for the entire operation, and (3) tons of ore and waste for the mining department.

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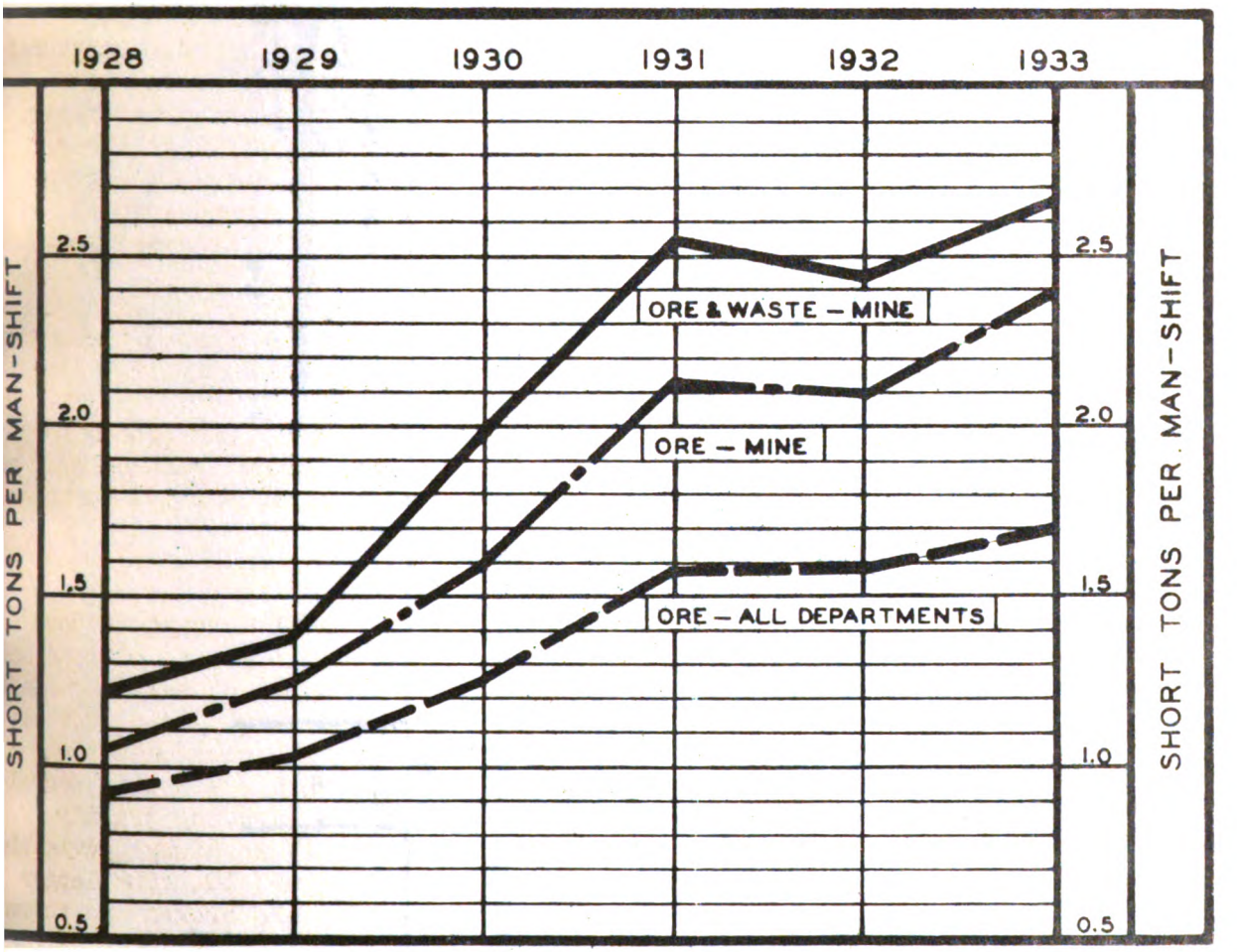

Figure 13.– Labor-efficiency record.

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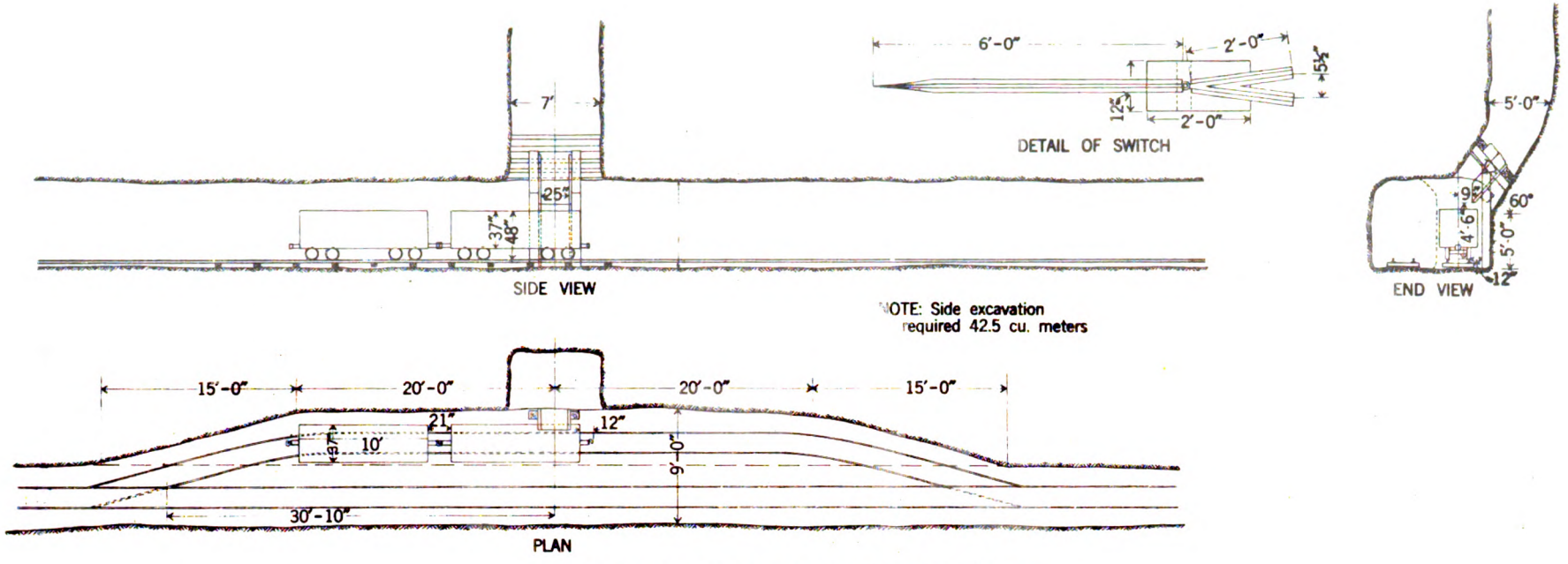

Figure 14.—Standard chute and turnout for 57-cubic-foot mine cars.

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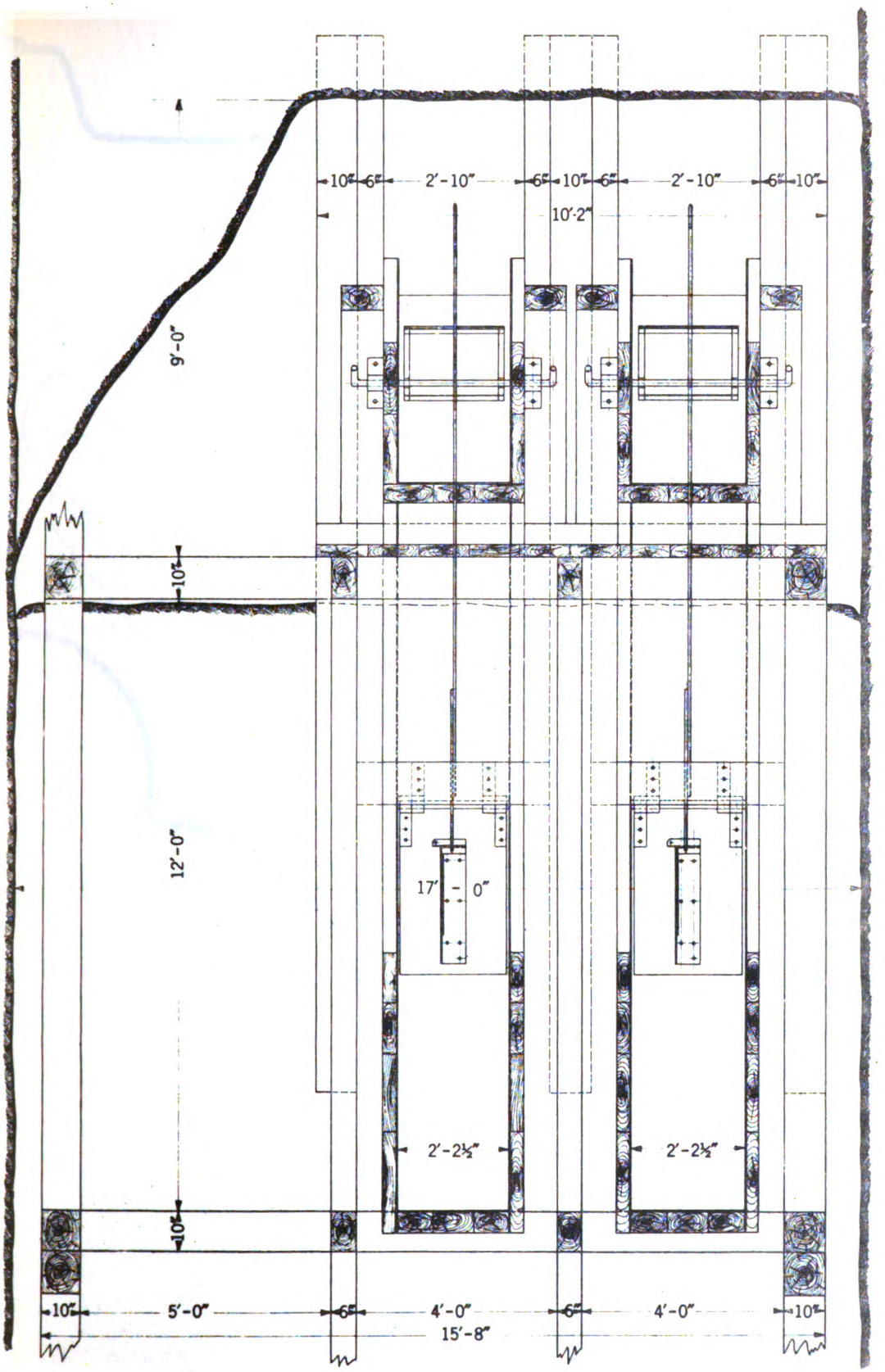

Figure 15.—Front elevation of skip-loading chute.

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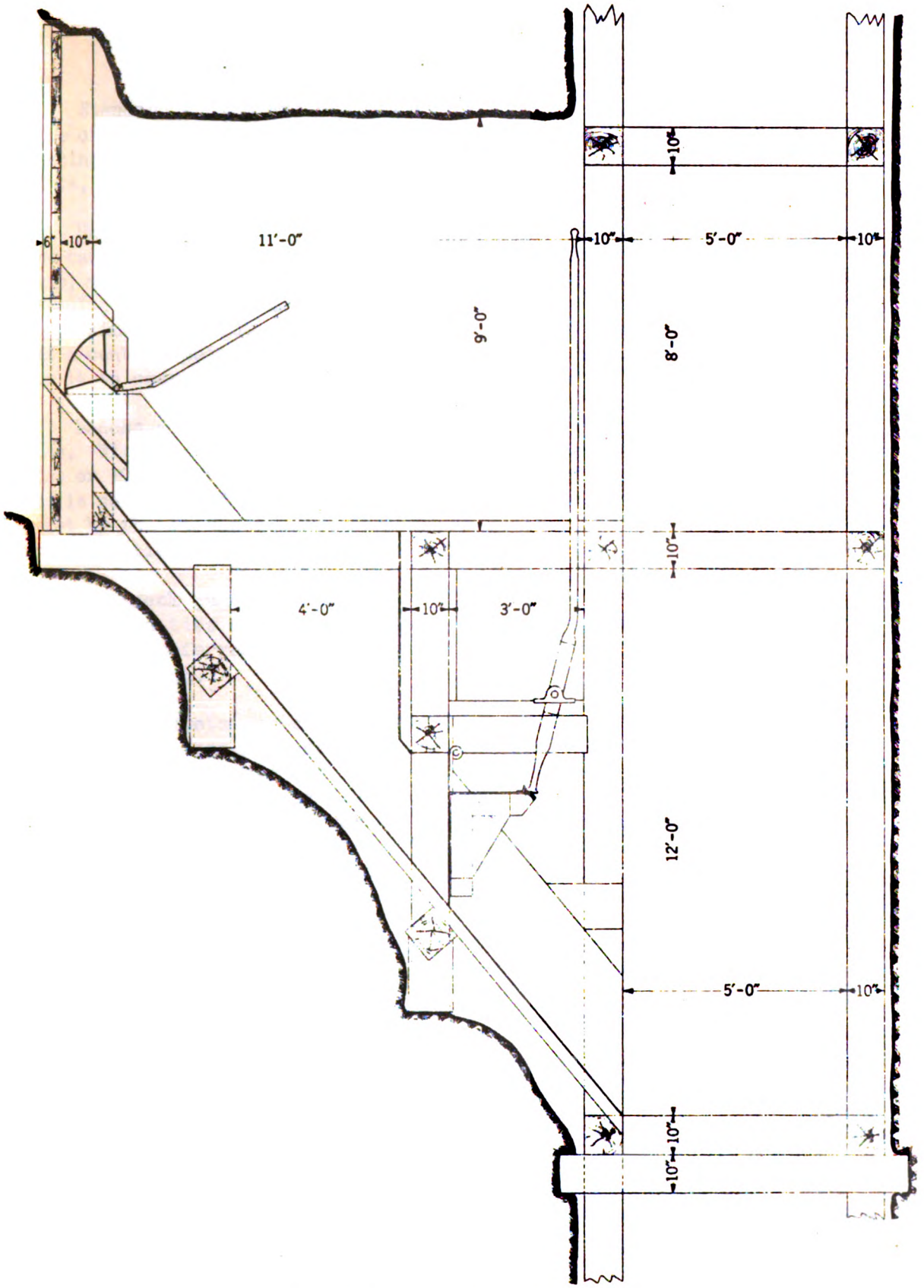

Figure 16.—Side elevation of skip-loading chute.

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UNDERGROUND HAULAGE

Standard chutes and chute gates are used in all workings. Chute gates are of the steel arc type, 24 inches wide, and are fitted into a 25-inch opening to allow free play. Enough side trackage is provided at loading points, as shown by figure 14.

Underground haulage equipment comprises 21-cubic-foot, end-dumping mine cars with Hyatt roller bearings, and 57-cubic-foot, V-bottom, side-dump, motor-haulage cars of the Park-Utah type, with Timken roller bearings. Haulage motors are Mancha trammers powered by Edison batteries and weigh approximately 3,000 pounds. The motor-generator charging set is located at a central point on level 18 of shaft no. 1 and branch feed lines run from it to charging racks on levels 16, 17, 18, and 19.

Mancha motors haul two 57-cubic-foot Park-Utah cars heaped full of sulphides. With the 21-cubic-foot cars a train usually comprises 6 cars of sulphides or 10 of carbonates. Trackage over the shaft pockets and in the stations is so arranged that the smaller cars may be dumped directly into the pocket by the station tender. The side-dump cars are run over a sloping grizzly and emptied much more rapidly than the small ones.

The track gage is 18 inches. Twenty-pound rails on 4- by 6-by 30-inch creosoted ties are normally used; where the traffic is heavy 30-pound rails are preferred. The tracks are properly ballasted. A simple latch switch is used.

Shaft pockets are cut on both sides of the shafts, one side being for waste and the other for ore storage. The ore pockets have a larger capacity than those for waste and hold 600 to 1,000 metric tons. Each waste pocket originally has a bypass from the manway compartment for handling muck from shaft-sinking operations, but as sinking goes deeper the bypass is eliminated. Ore is drawn from the pockets into a measuring pocket of 60-cubic-foot capacity and thence is discharged into a skip by a hand-operated device, as shown in figures 15 and 16.

Skips are hung above the cages and have a capacity of 70 cubic feet in no. 1 shaft and 50 cubic feet in no. 5 shaft. The average load of sulphide ore hoisted at no. 1 shaft is 4 dry metric tons and no. 5 shaft 3 dry metric tons.

Cages for transportation of men and material are double-decked; the upper deck may be hinged back to load timber, pipe or other long material. Folding safety gates protect front and rear entrances on both decks. Cages are made of light steel angles and perforated plate and have landing chairs underneath actuated by a lever inside the lower cage. The landing-chair mechanism is protected by steel bumpers to prevent damage when the cage is operating at or near the bottom of the shaft. Cages normally carry nine men on each deck. On main levels double loading platforms give access to both decks at once.

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HOISTING EQUIPMENT

At present the only active hoists are those at no. 1 and no. 5 shafts. The no. 1 hoist is an Allis-Chalmers double-drum unit driven by a 700-hp., 425 r.p.m., 550-volt, d.c. motor, which receives its power from a 600-kw., 1,200-ampere, 500-volt motor-generator set. The latter consists of a 3-phase, 60-cycle, 2,200-volt, 885 r.p.m. induction motor coupled to a compensated-shunt interpole generator. A 10-kw., 250-volt, continuous-duty exciter furnishes direct current to the fields of the hoist motor and generator and to the auxiliaries where required. A 32,000-pound, 10-foot-diameter, steel-plated flywheel is mounted on the shaft of the set to reduce the peak load on the service line.

The drums are 8 feet in diameter and 6 feet 6 inches wide, with grooved faces. The rope is a 6 by 19 lang-lay cable of 1-1/4-inch diameter. The hoist rating is 25,250 pounds pull on one rope, with a maximum hoisting speed of 1,500 feet per minute. All hoisting is done normally in balance. Hoisting capacity from an average depth of 2,200 feet is 80 metric tons per hour. The steel headframe has 7-foot-diameter, bicycle-type sheave wheels equipped with removable manganese-steel liners.

The no.5 hoist is an Allis-Chalmers double-drum unit driven by a 400-hp., 600-r.p.m., 550-ampere, 550-volt generator of the motor-generator set. The latter consists of a 3-phase, 60-cycle, 2200-volt, 1,175-r.p.m., 400-hp. induction motor coupled to a compensated-shunt interpole generator. A direct-connected, 9-kw., 250-volt, continuous-duty exciter furnishes direct current to the field of the hoist motor and generator and to the auxiliaries. A 25,000-pound, 8-foot-diameter, steel-plate flywheel is mounted on the shaft of the set. Drums are 7 feet in diameter, with grooved 6-foot faces, utilizing 6 by 19 lang-lay cables 1-1/8 inches in diameter. The hoist rating at the drum is 17,400 pounds' pull on one rope, with a maximum hoisting speed of 1,200 feet per minute. Normal hoisting is done in balance to an average depth of 2,000 feet; the capacity is 60 metric tons per hour. At this shaft the hoist-drum center line is only 103 feet from the center of the shaft because of limited space, and the headframe is 70 feet high. The fleet angle was so great that corrective measures were necessary to assure proper windings of the cables on the drums. Two auxiliary sheaves were set in the headframe, about 30 feet above the collar of the shaft and almost 9 feet apart. The ropes pass from the hoist drums under these sheaves, thence vertically up and over the main sheaves, which are set 9 feet apart on the hoist side and 4 feet apart on the shaft side--that is, exactly over the centers of the two hoisting compartments. Thus, the fleet angle is reduced greatly; the wear on the cables is not unreasonable, although it is higher than normal. The wheels are of bicycle type, 7 feet in diameter at the bottom of the groove, and equipped with removable manganese-steel liners.

Both hoists have multiple-arm friction clutches operated by independent auxiliary oil-actuated engines, liquid starters, and automatic slip regulators, Lilley safety controls, and Johnson graphical hoist recorders.

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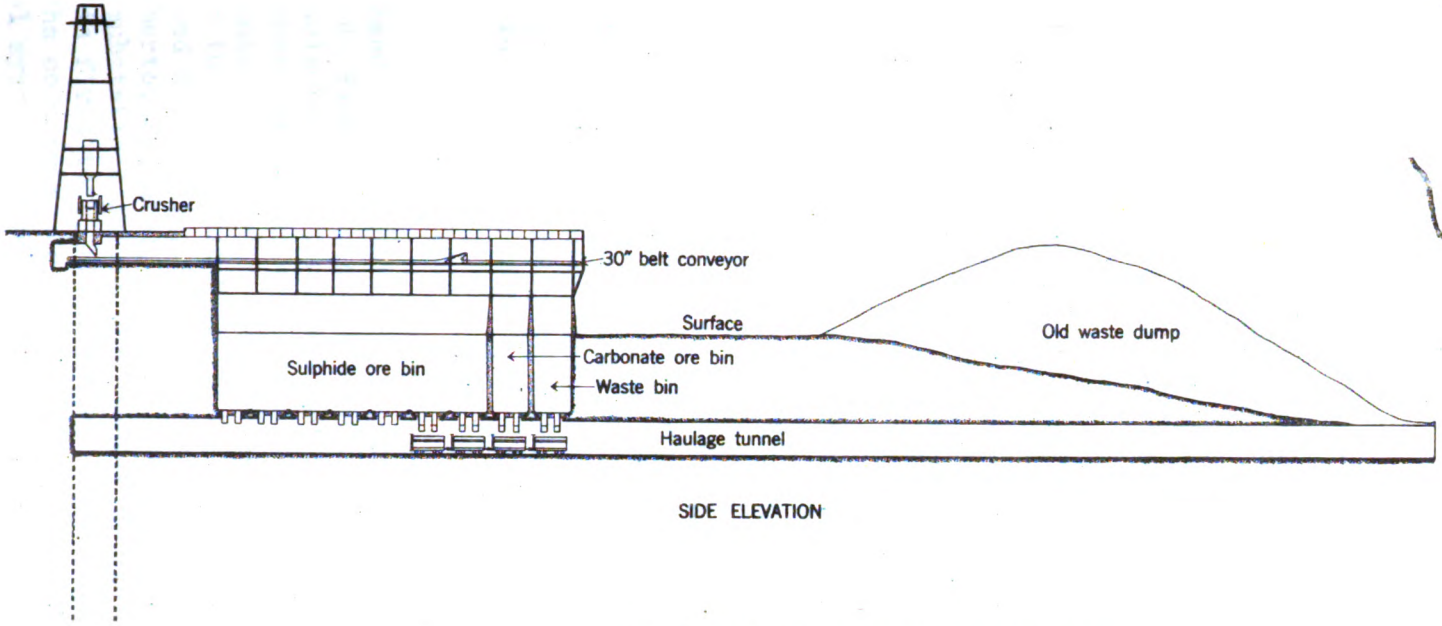
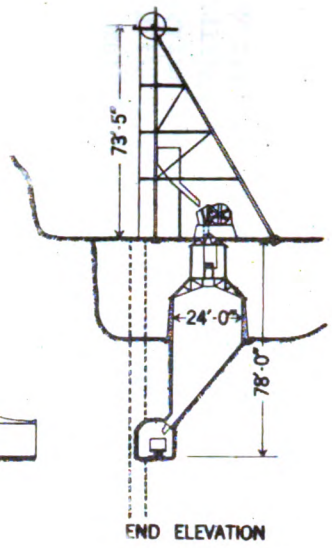
PLAN



SIDE ELEVATION



END ELEVATION

Figure 17.—Loading bins and tunnel layout at No. 1 shaft.

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The use of lang-lay rope on main hoists, although requiring more care in installation and handling, shows its superiority over regular-lay rope by its longer life. Service records of all ropes are kept in metric ton-kilometers, calculated on the basis of all up trips of ore and waste. In addition to following the manufacturers' recommendations as to primary lubrication, a secondary drip system is used on all main cable installations.

HANDLING ORE AND WASTE ON SURFACE

At no. 1 shaft ore and waste are dumped from the skips into a hopper in the headframe, from which they are fed into an Allis-Chalmers 36 by 4-inch Blake crusher set to 3 inches. They then pass onto a conveyor belt for distribution through a tripper into the reinforced-concrete receiving bins, which are partly above and partly underground (fig. 17). A belt directly beneath conveys the sorted waste to the waste compartment of the main storage bin. The storage bins have a capacity of 6,000 metric tons of sulfide milling ore.

Eighty feet below the shaft collar a railroad tunnel passes under the bins from the main patio level for 500 feet. Side-loading steel chutes discharge from the bins into 17-ton railroad cars, which are moved out to the main line by an electric trolley switching unit. This arrangement permits much easier loading, at the same time eliminating the bad grades and tortuous curves to the site of the old timber bins.

At no. 5 shaft waste is dumped into a receiving hopper in the headframe and then drawn into railroad cars and hauled away by the electric switching unit. Ore is dumped from the skip into a receiving hopper in the headframe, drawn into a railroad car and hauled by a Jeffrey trolley motor about three hundred feet to a receiving bin. From the latter a pan conveyor feeds the ore over a grizzly into an Allis-Chalmers Blake crusher similar to that at no. 1 shaft. In both installations the fines are by-passed on to the conveyor in order to cushion the belt and protect it from sharp rocks. A 4-inch conveyor belt with tripper distributes the ore into a 2,000-ton-capacity wooden flat-bottom bin with center bottom discharge gates. From the latter the ore is loaded into railroad cars.

POWER PLANT

All power is purchased from a subsidiary of the Electric Bond & Share Co., the Cia. Agricola de Fuerza Motriz del Rio Conchos, S.A. The main power lines come from a hydroelectric plant at La Boquilla, 98 miles south of Chihuahua; in addition, there is a stand-by steam plant at Torreon, Coah., about 275 miles south of Chihuahua. Power is transmitted over a 110,000-volt line from La Boquilla to the main substation at Avalos, 6 miles south of Chihuahua; there it is stepped down to 22,000 volts for transmission to Potosi substations at San Alberto, San Guillermo, and Santo Domingo. Company transformers at the several substations reduce the current to lower voltages; at Santo Domingo the change is first to 2,200 volts and then to 440, 220, and 110 volts as required. As the company power contract encourages reduction of power peaks, a remote-control system connects the substations to the central

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station at Santo Domingo, where an integrating wattmeter registers the load, together with the 15-minute power peaks. An ingenious, modern, electrical device limits the heavier power loads at peak times in the several hoisting and compressor plants. In addition individual charts showing the consumption of power by the principal electrical units aid in power analysis and the supervision of power distribution.

PERCENTAGE OF ORE EXTRACTION

The percentage of total ore recovery normally is high. The system of mining permits thorough final clean-up. Waste inclusions are seldom important except for their bearing on the calculation of ore reserves. A study of conditions in a given ore body permits fairly accurate deductions for waste ranging from nothing to 20 percent but usually about 10 percent.

WAGE AND CONTRACT SYSTEM

Of an average of 750 men at present (October 1933) employed in the mining and transportation department, 500 are classified as underground contract and days' pay labor, and the remainder comprise general surface, mechanical, electrical, and transportation personnel. Over a period of years a carefully controlled contract system has been developed in the mining department; and through it greatly increased effort is being obtained, with correspondingly lowered costs. Written contracts between native contractors and the company are made for all types of openings, and are in strict conformity with the Federal labor law. Unit prices for a given piece of work consider location and size of opening, character of formation, tramming methods (motor or hand), relative accessibility of the workings, and such other special factors as may govern. The company pays the wages of the contractor's men at weekly intervals, charging them to the contractor's account, together with his blasting supplies and carbide. Stoping-contract prices are based on 21-cubic-foot cars, 57-cubic-foot cars, or skips and exploration and development work on meters advanced or cubic meters excavated. Each contractor acts as a boss and is responsible for the proper execution of the work under him. Thus, comparatively little additional supervision is required. The contractor names machinemen as heads of the two shifts which he does not directly supervise.

Prices are adjusted when necessary to meet variables in stope conditions. The engineering department makes measurements twice a month. A detailed record of contracts is kept in the superintendent's office, including payments for ore, waste, advances, excavation, special work, total earned, shifts worked, price per car, price per meter, cars delivered, cars per man-shift, total meters driven, meters per man-shift, cubic meters excavated, supply cost per car, supply cost per meter of advance, supply cost per cubic meter excavated, amounts earned and guaranteed per man-shift, data on overdraw accounts, safety-record data, drill-equipment costs and production and assay values. This information is assembled for 15-day periods, most of it being obtained from the periodic contract liquidations. It makes available accurate information on costs for all types of mining work and serves not only for current control but as a basis for awarding new contracts.

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Results have entirely justified the detailed study and application of the system which, though somewhat complicated, has yeilded rich returns in low costs, efficient work, and a minimum of labor trouble.

The wage system provides standard payments for such work as it is not practicable to include in the contract system.

VENTILATION

In the main natural ventilation suffices. Some long headings advancing into new ground on the deep levels are ventilated temporarily by small, direct-connected electric fans and 12-inch canvas tubing. When connection is made with other workings these fans are dismantled for use elsewhere.

FIRE PROTECTION

Owing to the noninflammable character of the ore and the infrequent use of timber, danger from fire is slight. Fire rings connected to a high-pressure line are installed directly below the concrete collar of each shaft. On surface a high-pressure line connects to fire hydrants at strategic points. Furthermore, two chemical fire carts and numerous portable extinguishers are placed at salient points throughout the plants.

SAFETY AND HOSPITAL DEPARTMENT

A safety program gradually developed over a number of years has resulted primarily in a system of discipline and education. It was started with mass meetings for contractors, personal talks by foremen and shift bosses on their daily rounds, correction of unsafe features in operating, personal and committee conferences, and later compulsory suspension or expulsion from work for infractions of safety regulations. Use of the safety hat was increased gradually and eventually made compulsory, thus largely eliminating head injuries, some of which in prior years had resulted in fatalities. At periodic intervals a committee composed equally of workmen and company representatives meets to discuss all accidents since the previous meeting. At such time suggestive criticism is invited, which is an incentive to better safety practice.

Fresh posters conforming as nearly as possible to local conditions are placed on bulletin boards at regular intervals. The men are given periodical physical examinations; in addition, any workman who suffers a lost-time accident is re-examined to determine whether there is anything in his physical condition which might have contributed to his accident.

The safety department normally comprises a safety director who also acts as employment agent and an office assistant who prepares posters and helps maintain adequate records. On each shift underground trained first-aid men treat all minor injuries at a first-aid station, sending those of a

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more serious nature to the company hospital on surface. They also make necessary dressings on the surface at the beginning and end of each shift. These men work in cooperation with and directly under the supervision of the medical and safety departments. The introduction of first aid underground has done much to reduce lost-time accidents and minimize infections from superficial cuts and bruises.

The company maintains an adequate, modern, industrial hospital. The staff consists of a head doctor and assistant doctor, trained nurse, and record clerk. Equipment consists of a six-bed ward, laboratory, office drug-dispensing room, first-aid dressing room, operating room, X-ray and dark room, and private room for isolation cases. A 100-milliampere, G.E. X-ray machine is utilized for examining any workmen who require this diagnostic aid.

Safety data since 1927 have indicated steady improvement; there were 3.184 accidents per 1,000 shifts in that year, 0.088 in 1932, and 0.3[illegible] to October 1, 1933. Shifts lost per disabling accident were 40.050 in 192[illegible] 16.798 in 1932, and 5.540 to October 1, 1933. These statistics include fatalities, which are charged as 612 shifts lost per fatality. Figure 16 shows graphically the progress of the safety program.

SUPPLY DEPARTMENT

The supply-department organization consists of a purchasing agent in the United States, a local purchasing agent at the general office in Chihuahua, and the warehouse personnel.

The mine warehouse is a modern, fireproof structure, designed for utmost visibility and accessibility according to a standardized scheme similar to that employed by several large railway companies and industrial enterprises in the United States. The building is approximately 150 feet long 40 feet wide. With a few exceptions unit piling is used. Such items as bo rivets, nuts, and nails are placed in trays of unit weight and stacked in their proper compartments. Material required in less than unit amounts is weighed out from the top tray. If at any time inventory is required for a given item, it is only necessary to count the lower trays, multiply by the unit capacity, and add the weight of the material in the top tray. Oils a greases are stored in a special warehouse oil room. General-purpose oils kept in individual tanks filled from the outside of the building. Greases and paint are stored in small containers.

Card-index records are maintained for all items to show their ra of movement. Every effort is made to prevent overstocking. In many cases, substitution of slow-moving items for others has aided materially in lower inventories.

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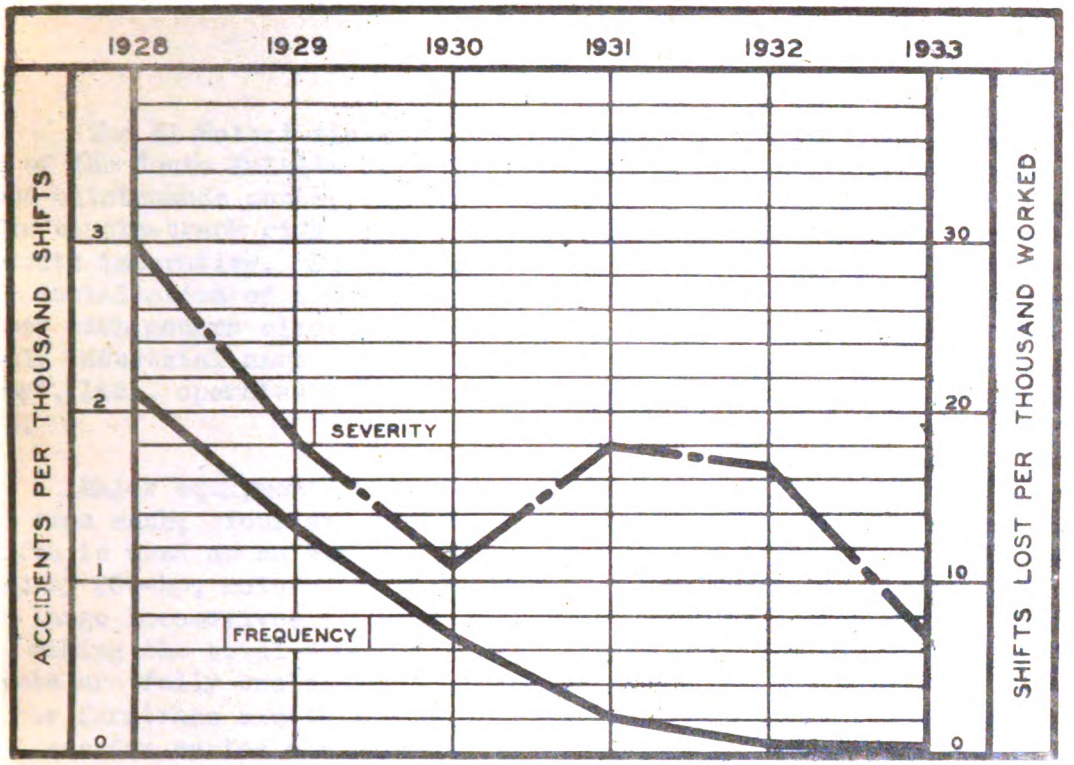

Figure 18.—Accident record, 1928-33.

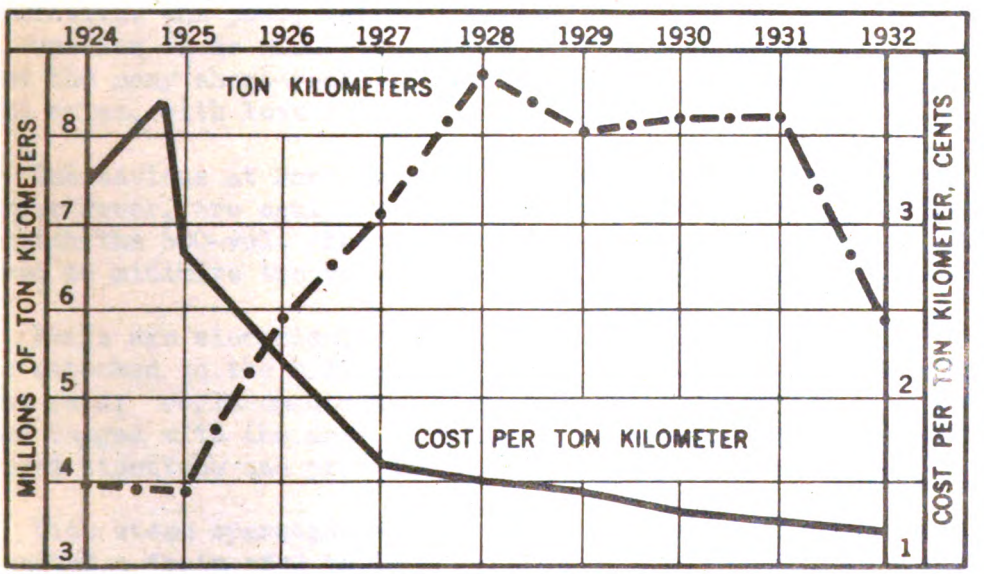

Figure 19.—Railroad costs and tonnage.

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TRANSPORTATION

The El Potosi Mining Co. 30-inch-gage steam railroad served the mines of the Santa Eulalia district for many years; but increasing operating and maintenance costs, together with congestion of traffic over the 15-mile single-track right-of-way between the mines and smelter, made improvements imperative. During 1925 the entire main line was electrified by the installation of a 600-volt, direct-current, overhead trolley system, together with modern electric locomotive and substation equipment. This is the only industrial narrow-gage electric railway in Mexico; the Mexican Railways, Ltd., operates a 70-mile broad-gage electric road in southern Mexico.

Major equipment includes five General Electric locomotives weighing 25 tons each; four are coupled in pairs as double or tandem units, and the fifth is used as an extra. Two 13-ton General Electric switching units, powered by 200-hp. motors, make up trains at San Guillermo and Santo Domingo. In the large locomotives a 200-hp. motor is mounted on each double-swivel truck, making the total 400-hp. to each single unit and 800 in tandem. Locomotive cabs are fully enclosed; a pantograph-type trolley is used. A motor generator furnishes excitation to the motors when acting as generators, with connections for series and parallel braking, permitting braking speeds between 8 and 20 miles per hour.

One double unit is used for transportation between San Guillermo, the Avalos smelter, and the lower terminal of the system at Hacienda Robinson. The other unit is employed for transportation of ore and supplies between San Guillermo and Santo Domingo, with occasional trips to the smelter.

Although regenerative braking is one feature in the design of the large locomotives the power returned to the line has secondary importance; its main function is as a safety feature in handling trains down heavy grades and around the many short-radius curves. It also diminishes the strain on wheels and axles, with less resultant breakage.

Substations at San Alberto, San Guillermo, and Santo Domingo, mentioned under Power, are equipped with 500-kw., synchronous motor generators, which furnish the 600-volt direct current to the line. A double-feed line is employed to minimize the voltage drop between substations.

Rails are electrically bonded with 2/0 copper bonds, with steel terminals attached to the ball of the rail. Rail sizes range from 35 to 50 pounds per yard; replacements are made with 50-pound rail. Ties are creosoted and tagged with the date of their replacement. Track is ballasted with crushed limestone and provided with ample drainage.

With steam operation a crew of 5 - engineer, fireman , and 3 brakemen - handled a train of 8 to 16 cars, transporting a maximum of 120 short tons of ore down grade to the smelter. A crew of the same size now mans a double unit, hauling 250 short tons of ore from mine to mill. Under steam

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operation seven train crews were required daily to handle a monthly production of 25,000 short tons of shipping ore. With electrification, the personnel has been reduced to 3 double-unit shifts and 2 switching units shifts per operating day to handle 50,000 tons of mill sulphides and all concentrate shipments.

Major benefits of railway electrification are as follows:

1. Marked increase of train loads per operating crew resulting in corresponding decreases in labor and supply costs.

2. Increased daily tonnage capacity.

3. Ability to handle normal high tonnages, which, with the greater congestion resulting from the use of steam trains, would have been impossible.

4. Decreased maintenance costs, fewer accidents and derailments from broken wheels and axles and reduced running time per trip.

Marked economies noted in the cost tabulation below are to be credited in part to greater tonnage volume. Still further traffic increases are possible by the simple expedient of operating more shifts, whereas it would be out of the question to meet current demands with steam equipment over the single track right-of-way (see fig. 19).

Operating Costs of El Potosi Railroad

by years, 1924 - 32 1/

Year	Cost per short ton	Cost per short ton - mile	Short tons moved
1924	$ 0.705	$ 0.0466	178,580
1925	.404	.0404	228,741
1926	.290	.0314	436,806
1927	.193	.0227	581,873
1928	.187	.0217	696,633
1929	.169	.0209	684,086
1930	.150	.0193	724,044
1931	.130	.0183	794,202
1932	.125	.0179	577,691

1/ All 1924 and 4 months of 1925 were on steam operation. Remainder was electric operation. Labor and supply items involving the expenditure of Mexican currency are all calculated into the above costs at an exchange rate of 2 to 1.

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MINING COSTS

Mining costs in dollars (U.S. currency) and in units of labor, power, and supplies are given in tables 2, 3, 4, and 5.

Power consumption is shown by years for 1928 to 1932 in figure 20 A and stoping costs for 1927 to 1932 in figure 20 B.

Drill-steel consumption by years is given in table 6.

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TABLE 2.- Summary of costs, El Potosi Mining Co., August 1933.

Tons of ore mined and hoisted - 53,500 (short tons).

Mining method - underhand stoping, open stopes with pillar support.

	Underground costs per ton mined.							
Development	Labor	Super-vision	Comp.air, drills, steel	Power	Explo-sives	Timber	All other	Total
In ore	$0.027	—	$0.011	$0.007	$0.012	$0.002	-	$0.059
In rock (limestone)	.047	-	.015	.015	.018	.001	$0.008	.104
Diamond drilling	.018	$0.006	-	.006	-	-	.009	.039
Stoping (ore breaking)	.180	-	.009	.058	.052	.010	.028	.337
Transportation (underground)	.010	-	-	.002	-	-	.008	.020
Drainage	.001	-	-	.001	-	-	-	.002
Hoisting	.010	-	-	.059	-	-	.011	.080
General underground expense								
Track	.003	-	-	-	-	-	.002	.005
General	.013	.042	-	-	-	-	.005	.060
General surface expense	-	-	-	-	-	-	-	.143
Grand total	$ .309	.048	.035	.148	.082	.013	.071	.849

Note: Supervision shown under General underground expense.

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TABLE 3.- Summary of costs in units of labor, power and supplies
El Potosi Mining Company and Buena Tierra lease

s of ore and waste produced - 62,507 (short tons)
ing method - underhand stoping, open stopes with pillar support.

	Man-hours	Tons per man shift	
Labor (Man-hours per ton)			
Mining (stoping and development combined)	1.410		
Timberman	.031		
Hoisting and haulage	.138		
Supervision	.066		
Total underground	1.645	4.863	
Diamond drilling	.064		
Mechanical department	.268		
Electrical	.057		
General surface	.338		
Mine staff	.097		
Railroad	.177		
All other	1.001	7.990	
Grand total	2.646	3.023	
Labor and supervision percent of total cost			42.05

	Development	Mining	Total
Power and supplies			
Explosives (lbs. per ton) 35% strength	0.219	0.384	0.603
Total power, kw.-hr. (development and mining combined)			
Air compression	-	3.61	3.61
Hoisting	-	3.75	3.75
Pumping	-	.15	.15
Ventilation	-	.03	.03
Lighting	-	.19	.19
Diamond drill	-	.02	.02
Crushing (primary)	-	.34	.34
All others	-	.14	.14
Total kw.-hr.	-	8.23	8.23
Other supplies in percent of total supplies and power			69.92
Power - percent of total cost			17.43
Supplies and power - percent of total cost			57.95
Percent of total cost	23.8	76.2	100.00
Other supplies in percent of total cost			40.52

[1] In calculating percentages under C, total development is taken from table 1 as $0.202 and compared with grand total of $0.849.

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TABLE 4. - Detailed development costs in units of labor, power and supplies

Size of excavation - 5 by 7 feet.

No timber

Physical properties of rock - soft limestone, easy to drill, but very tenacious and difficult to blast.

Labor, man-hours per foot	3.49
Feet per man-shift (per 8 hours)	2.30
Labor, percent of total cost	45.5
Explosives (lb. per foot)	8.21
Power, percent of total cost	13.9
Power and supplies, percent of total cost	54.5

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Period covered - years 1926 - 32 and August 1933.

Mining method - underhand stoping

Ore - silver-lead-zinc milling sulphides.

	1926	1927	1928	1929	1930	1931	1932	Aug. 1933
Ore breaking and general underground expense	$1.275	$1.046	$1.107	$1.035	$0.703	$0.520	$0.425	$0.402
Hoisting	.200	.117	.133	.149	.108	.106	.087	.080
Tramming	.035	.038	.044	.047	.031	.028	.023	.020
Drainage	.034	.015	.010	.019	.010	.005	.004	.002
Indirect costs	.760	.401	.408	.375	.232	.199	.203	.143
Development and diamond drilling	.736	.646	.748	.427	.503	.453	.269	.202
Total mining	$3.040	2.263	2.450	2.052	1.637	1.311	1.011	.849
Railroad	.297	.203	.199	.187	.175	.133	.129	.111
Total mining and railroad	$3.337	2.466	2.649	2.239	1.812	1.444	1.140	.960
Short tons mined	121,848	250,316	316,840	399,518	424,363	552,923	441,094	53,500

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TABLE 6.- Drill-steel consumption, by years.

	1926	1927	1928	1929	1930	1931	1932	1933
Consumption, lb.	76,441	71,672	66,330	52,128	38,233	24,811	14,522	18,770
Consumption, lb. per ton	0.316	0.179	0.130	0.099	0.061	0.034	0.027	0.039
Air-drill shifts	27,943	49,966	65,148	57,763	45,819	44,121	29,763	22,474
Pieces steel sharpened	104,127	314,793	252,208	174,584	119,782	82,665	56,798	51,922
Pieces sharpened per ton	0.430	0.786	0.496	0.332	0.191	0.113	0.108	0.109
Tonnage (ore and waste)	241,972	400,307	508,458	526,420	626,333	731,184	528,165	478,285

Note: In 1932 operations were on 5-day basis until August, closed until September 17, then on a 3-day basis until August 1933, when 5-day basis was resumed. 1933 is for 11 momths only. Tonnage includes milling ores, shipping ores, and waste produced from development.

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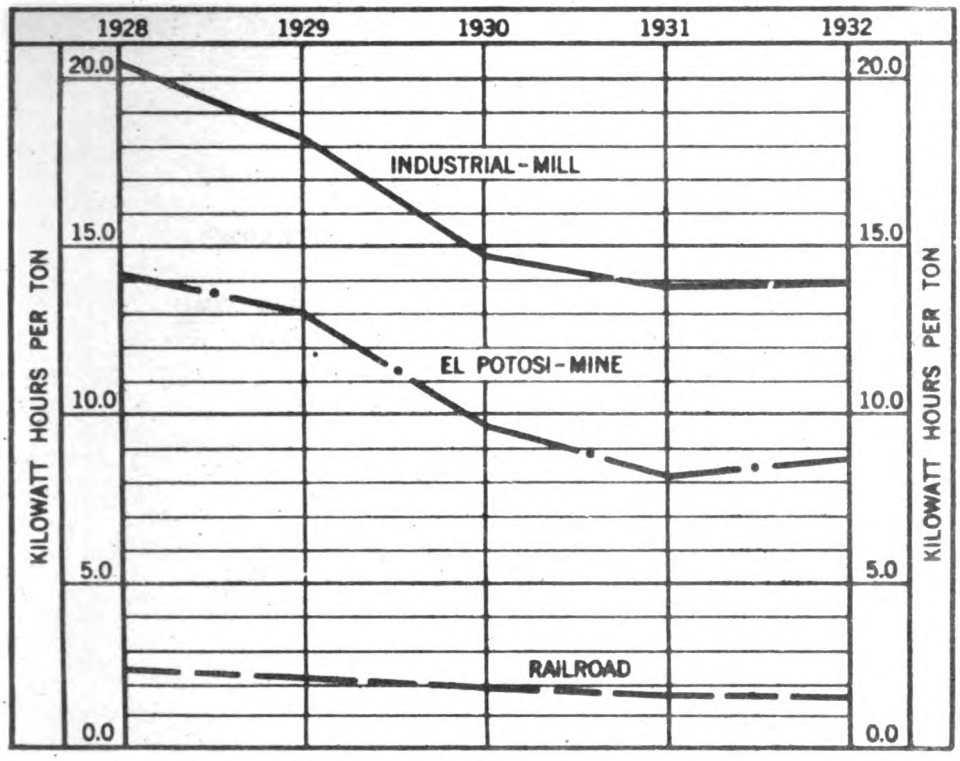

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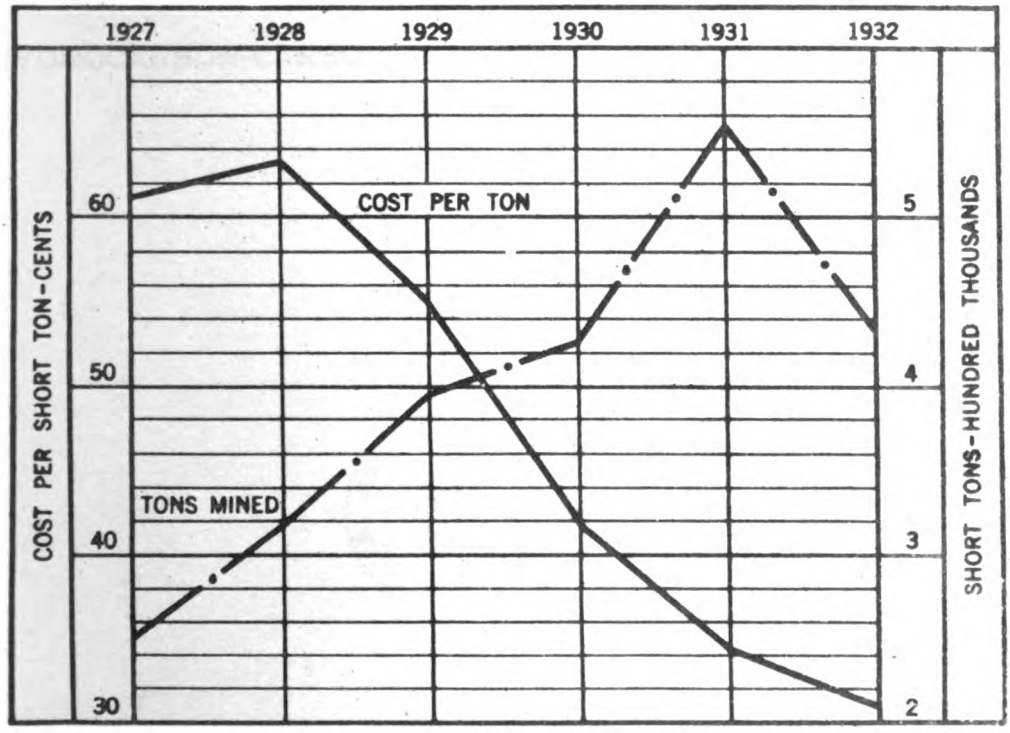

B

Figure 20.— Power consumption and stoping costs: *A*, Electric power consumption, by years, 1928-32; *B*, mill-ore stoping costs, by years, 1927-32.

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